PREFACE

TO THE FIRST EDITION.

THE collection of papers which forms this book was mostly prepared in moments stolen from more active professional duties, and must consequently lack the uniformity and completeness which is compatible only with ample leisure and freedom from other more pressing cares.

It has been my intention to confine myself principally to facts gleaned from my own experience, and only to touch upon theoretical questions when essential for the understanding of practical facts.

As the items of cost, both of construction and subsequent operation, are amongst the most important of all the practical questions that face the originators of new smelting enterprises, and as these are virtually unattainable to the general public, I have gone into these figures in considerable detail, not calculating expenses as they appear on paper, and when everything is running smoothly, but giving the actual results of building on a large scale, and of smelting many thousand tons of ores under varying circumstances, and in all of the ordinary kinds of furnaces.

Owing to the magnitude of the subject, I found it impossible to touch upon the so-called "Wet Methods" without increasing the size and consequent cost of this volume to an extent that might probably peril its circulation.

The author desires to acknowledge the valuable assistance of Mr. J. E. Mills, in connection with the geology of the Butte mining district, and to credit Mr. H. M. Howe and Mr. A. F. Wendt with the use he has made of their papers on "Copper Smelting" and on "The Pyrites Deposits of the Alleghanies."

But, above all, he has to thank Mr. James Douglas for a thorough and minute revision and criticism of his manuscript just before publication.

E. D. P., JR.

WALPOLE, MASS., June, 1887.

— 6-18507 —

PREFACE

TO THE SECOND EDITION.

THE time which has elapsed since the exhaustion of the large first edition of this book has given an opportunity for its thorough revision, as well as for the addition of such new material as time and experience have suggested. The advances in copper smelting since this work first appeared have been rather in a general enlargement of furnaces and apparatus than in any radical changes or inventions.

Amongst many friends who have given their valuable assistance I desire to mention, in particular, Mr. Francis L. Sperry, of Sudbury, Ontario, for five years analytical chemist for the Canadian Copper Co., who has prepared the section on the Electrolytic Assay of Copper, and Mr. Paul Johnson, of Sweden, who has furnished me valuable information and plans of the regenerative gas furnaces for copper refining now used at Atvidaberg, Sweden. I have ventured to introduce this description of foreign practice in the belief that it is to the use of regenerative gas furnaces that we must look for any great economy in fuel, a point absolutely vital in some of our Western mining districts.

E. D. P., JR.

NEW YORK, November, 1891.

PREFACE

TO THE THIRD EDITION.

It is scarcely a month since I appended my initials to the Preface to the Second Edition of this book.

The sheets of that edition had barely time to lose the moisture from the operation of printing before they were exhausted.

Consequently I have no changes to make in the text, and nothing to offer as a preface, except my sincere thanks to the Public at Large, and to my Brother-Metallurgists in particular, for the extreme kindness with which they have received this record of personal experience.

Amongst some thirty or forty elaborate reviews and criticisms of this work, published in magazines and journals scattered all over the world, I have not seen a single word of fault-finding; while I have felt almost overwhelmed by the unanimous praise that they have independently bestowed upon it.

It is an ample reward for the great labor and time that must be expended by whomever undertakes to write a technical book.

E. D. P., Jr.

Boston, December, 1891.

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MODERN AMERICAN METHODS
OF
COPPER SMELTING.

CHAPTER I.

DESCRIPTION OF THE ORES OF COPPER.

ALTHOUGH the copper-bearing minerals are numerous, yet those of commercial importance are few in number, and for the most part quite simple in chemical composition. The following minerals may be properly considered ores of copper, and are found in the United States in the localities enumerated.

NATIVE METALLIC COPPER.

Aside from the extensive occurrence of this metal in the Lake Superior region and at Santa Rita, New Mexico, it is found very frequently as a product of decomposition, though seldom in sufficient quantities to render it of any commercial importance. It is usually remarkable for its purity.

CUPRITE, OR RED OXIDE OF COPPER, Cu_2O; 88·8 Cu, 11·2 O.

This mineral occurs solely as a product of decomposition, and while quite widely distributed, is nowhere an ore of any importance, except in the Southwestern carbonate mines, where it sometimes permeates large masses of iron oxide, notably increasing their copper contents. Quite large lumps of this mineral are found in the Santa Rita mines, and are evidently the result of an oxidation of nodules of metallic copper, the unaltered center being usually preserved of greater or less size.* Many of the Butte City veins, as well as fissures throughout the Eastern Coast Range, carry this mineral in their upper portions as a product of the decomposition of sulphide ores.

MELACONITE, BLACK OXIDE OF COPPER, CuO_2; 79·8 Cu, 20·2 O.

This ore, with its metallic contents usually in part replaced by oxides of iron and manganese, is not quite so widely distributed as the sub-oxide, but is more frequently found in masses sufficiently large to pay for extraction. Its most remarkable occurrence in the United States was in the Blue Ridge mines of Tennessee, North Carolina, and Virginia, where the upper portion of the beds furnished a very large amount of from 20 to 50 per cent. of ore, having the appearance of melaconite, and giving rise to expectations that were always shattered after passing through this rich zone and reaching the lean, unaltered pyrites below. This so-called black oxide of the Blue Ridge region † seems to be an intimate mixture of glance, oxide, carbonate, and sometimes finely divided native copper. Two analyses, by Dr. A. Trippel, show their constituents:

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* An average sample of thirteen tons of concentrates, taken by the author at Santa Rita, in 1881, and partially analyzed under his supervision, gave, after continuing the concentration by hand to almost complete removal of the rock constituents:

Oxides of copper	13·42
Carbonates of copper	1·27
Oxides of iron	0·13
Metallic iron (from stamps)	0·29
Sulphur	0·11
Insoluble residue	0·37
Metallic copper	83·66
Zn, Ag, Co, Ni, Pb, Mn	Traces
	99.25

This analysis presented points of considerable difficulty, especially in determining the amount of oxide of copper in the presence of metallic copper. Entirely satisfactory results were not obtained; but the method proposed by W. Hampe, by means of nitrate of silver, yielded the only figures that could lay the slightest claims to accuracy.

† *Pyrites Deposits of the Alleghanies,* by A. F. Wendt.

Oxide of copper	5·75	3·80
Sesquioxide of iron	1·50	·63
Sulphur	18·75	25·40
Copper	71.91	41·00
Iron	.93	26·56
Soluble sulphates of copper and iron	·72	1·78
	99·56	99·17

A pile of such ore, laid on a bed of cordwood and moistened, often ignites the wood below, and thus roasts itself without firing.

Malachite, Cu_2O, $CO_2 + HO$; 71·9 CuO, 19·9 CO_2, 8·2 HO.

This is a much more valuable compound of copper than the two preceding oxides, from a commercial standpoint; although no mines in the United States furnish malachite of sufficient purity to fit it for ornamental purposes.

While it may be said to occur in widely distributed but ordinarily in non-paying quantities, in the upper decomposed regions of most copper deposits, there are certain localities in which it forms the principal ore of this metal. It is very seldom found in a state of purity, but is mixed with various salts of lime and magnesia, oxides of iron and manganese, silica in its various forms of quartz, chalcedony, flint, chert, and jasper, and when seemingly present in large quantities, it often forms only worthless incrustations, or merely colors green nodules and masses of valueless material. It is then difficult, and in some cases impossible, to form any accurate opinion of the tenor of the ore from its external appearance.

Azurite, $2CuO$, $CO_2 + HO$; 69·2 CuO, 25·6 CO_2, 5·2 HO.

This mineral requires only a passing notice. It is distributed in the same manner and occurs under the same conditions as its sister carbonate, but in very much smaller amounts. It occurs in profitable quantities only in some of the Southwestern mines. Specimens of this mineral are found with malachite and calc-spar in the Longfellow mine, exceeding in beauty anything of the kind that is known elsewhere in the United States.

Chalcopyrite, Cu_2S, Fe_2S_3; 34·4 Cu, 30·5 Fe, 35·1 S.

This is by far the most widely distributed ore of this metal, and furnishes the greater proportion of the world's copper. It occurs principally in the older crystalline rocks, frequently accompanied with an overwhelming percentage of iron pyrites, in bedded veins in Newfoundland, in Quebec, Canada, in Vermont, Virginia, Georgia, Tennessee, and Alabama.

The value of copper-bearing fissure veins below the limit of surface decomposition is nearly always due to this mineral. In some localities the chalcopyrite forms with pyrite a fine-grained mechanical mixture, varying in color with its percentage of copper from deep yellow to steel-gray. This substance is easily recognized under the microscope as a mechanical mixture, and not a chemical compound. In most of the carbonate mines of the Southwest that have attained any considerable depth, chalcopyrite is already becoming apparent, in minute specks; and it is highly probable that the altered ores near the surface, with their valuable admixture of ferric oxides, are all due to the decomposition of this mineral. The sulphureted fissure-veins of the Rocky Mountains and Sierra Nevada are seldom free from this mineral, although their value almost invariably depends upon their precious metal contents. The remarkable purple ores and copper glance of Butte City, Montana, have already in several mines given place in depth to the universal yellow sulphide.

CHALCOCITE, COPPER GLANCE, Cu_2S; 79·7 CU, 20·3 S.

This ore is seldom found in a condition of perfect purity, its valuable component being frequently in part replaced by iron and other metals. Its copper percentage rarely falls below 55, and even at this low standard the mineral retains its physical characteristics, a slight diminution in its luster being the principal difference observable. When high in copper, it greatly resembles the white metal of the smelter. Chalcocite containing from 60 to 74 per cent. of copper occurred also pure and is relied on in the noted Anaconda mine, Butte, Montana. Several of the other Butte mines carry the same mineral, although, as they approach the western boundaries of the district, it gradually passes into bornite or peacock ore. It is also an important ore in Arizona, occurring in large quantities near Prescott, as well as in the Coronado and other Clifton mines. In New Mexico, it constitutes virtually the entire value of the Nacimiento and Oscura Permian beds. It occurs frequently in Texas in the Grand Belt mines, and is the principal ore of numerous narrow fissures in the Middle and Atlantic States. In the Orange Mountains of New Jersey, examined by the author, it was found in a species of shale, as an ore of the following composition:

Copper	75·20	Sulphur	17·97
Iron	4·10	Insoluble	1·10
Manganese	1·13		
Silver (2·37 ounces)	0·01		99·51
Gold	Trace		

BORNITE OR ERUBESCITE, $3Cu_2S, Fe_2S_3$; 55·58 CU, 16·36 FE, 28·06 S.

This is one of the most beautiful of the sulphureted ores of copper, being characterized in its fresh condition by a superb purplish-brown color, which soon changes on exposure to the air into every conceivable hue, from a golden yellow to the deepest indigo, and from a brilliant green to a royal purple. The mode of occurrence of this mineral and its limited extent of distribution as regards depth indubitably stamp it as a product of decomposition, solution, and re-deposition of the metallic portion of the vein. Like copper glance, this mineral is far from uniform in its composition, varying in richness from 42 to nearly 70 per cent. of copper without entirely losing its characteristic colors.

It forms one of the principal ores of the Butte mines in their rich zone, which lies between the leached-out surface zone and the unaltered low-grade sulphides below.

TETRAHEDRITE, GRAY COPPER ORE, FAHLORE $(Cu_2S, FeS, ZnS, AgS, PbS)_4 (Sb_2S_3, As_2 S_3)$; 30·40 PER CENT. COPPER.

Except in those rare and highly argentiferous varieties in which the copper is replaced to a greater or less extent by silver, this is seldom regarded in the United States as an ore of copper.

Both its scarcity and its obnoxious components (arsenic, antimony, etc.) prevent its use as a source of copper in this country, where the extreme purity of our ores has established such a high standard for this metal. Only the most favorable circumstances, mineralogical, metallurgical, and commercial, would render the working of non-argentiferous fahlores at all practicable. This mineral occurs in small quantities in certain of the Butte copper mines, rendering their product slightly inferior to that from the oxidized ores of Arizona or the pure sulphides of Vermont. This slight disadvantage is, however, far outweighted by their contents in silver, which doubtless owes its presence to this same arsenical mineral. From the San Juan region, Colorado, an argentiferous tetrahedrite adds a notable quantity to the production of the United States. It appears principally as matte from the lead furnaces, and as black oxide from the Argo separating works.

CHAPTER II.

DISTRIBUTION OF THE ORES OF COPPER.

THE ores of copper are widely distributed over the earth's surface, and may be found in almost every geological formation; but the deposits of commercial importance had their origin, for the most part, at a very early period of the world's history. The mines of the old world, as well as those of Chili and Australia, having been described with great minuteness by various careful authors, there remain to be mentioned in this place only the principal copper districts of North America, which, for convenience of description, may be classed in four groups:

I. The Atlantic coast beds.
II. The Lake Superior deposits.
III. The mountain system of veins.
IV. The Southern carbonate deposits.

I. THE ATLANTIC COAST BEDS.

Throughout its whole extent in North America, the Atlantic coast is bordered by a succession of parallel ranges, which, by their general geological as well as geographical analogy, must be classed in the same system. They form an unbroken chain from Florida to Labrador, and thence, continuing their same northeasterly direction along the coast of that bleak country, dip beneath the waters of Baffin's Bay, where they are represented by a series of submarine peaks, and, nourishing the gigantic glacier system of Western Greenland,* terminate, so far as known, in Mount Edward Parry, north latitude 82° 40′. Dr. T. Sterry Hunt's admirable researches have given us a very clear insight into the origin, formation, and structure of this immense range of mountains within the confines of the United States and Canada. It consists essentially of metamorphic rocks—largely crystalline schists—and is metal-bearing to a greater or less degree throughout its entire extent, though only in a few places is copper found in a sufficiently concentrated form to justify any attempts at extraction.

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* See Dr. Kane's *Arctic Expedition* for soundings taken in Baffin's Bay; also *Geology of Greenland's Mountains.*

The only copper mineral of importance in this range is chalcopyrite. In the more northerly division, where there has been extensive glacial denudation, this reaches unaltered almost, or quite, to the grass-roots, while from Virginia to Tennessee, where abrasion has not taken place, and where oxidation has been assisted by climatic influences, decomposition with subsequent concentration is found to a considerable depth. The result of this is usually a zone, rich in an impure black oxide of copper containing a certain proportion of sulphur, which sometimes occurs in considerable quantities near the surface, after first passing through a greater or less extent of barren iron oxide, derived from pyrite, and which has no doubt furnished the copper to enrich the underlying zone.

The occurrence of this valuable mineral in merchantable quantities has, in more than one instance, raised expectations and led to large expenditures that have subsequently proved entirely unwarranted; for at a slightly greater depth, the unaltered vein assumes its true character of a more or less solid pyrite and pyrrhotite, carrying a very small amount of copper (seldom above three per cent.) in the form of the common yellow sulphuret. When the accompanying mineral is a bisulphide of iron and the locality is favorable, the pyrite may be utilized in the manufacture of sulphuric acid, the copper being extracted from the residues by well-known methods; but when the prevailing mineral is the *mono*sulphide—magnetic pyrites—there can be no question of profitable working, pyrrhotite being absolutely valueless since copperas has become a by-product of fence-wire making. At Capelton, in Canada, at Ely, Vermont, and at one or two points in Newfoundland, copper pyrite occurs in a sufficiently concentrated form to yield from five to six per cent. in considerable quantities, an ore on which profitable operations may be conducted, under favorable conditions.

In Virginia, at Ore Knob, North Carolina, at the Tallapoosa mine, in Georgia, and at Stone Hill, Alabama, indications of a similar concentration of copper have given rise to extensive explorations, and, in some cases, to the expenditures of large amounts of money, which have not always resulted satisfactorily. These are all examples of so-called *bedded veins*, following the lines of stratification, and being simply sandwiched in between the layers of rock. One of the most curious features of these beds is the alternate occurrence of the sulphide of iron that forms the great mass of the gangue, as pyrrhotite and pyrite. In Capelton, for instance, we have the bisulphide; a hundred miles distant, at Ely, the monosulphide alone exists; in Virginia and at Ore Knob, the monosulphide preponderates; while in the Tallapoosa mine, the bisulphide alone is found. Neither the chemical nor geological composition of the corresponding country-rock explains this phenomenon. Here, it will be proper to mention the occurrence, in stratified rocks, of the sulphide of copper (copper glance), usually in unimportant quantity, throughout Pennsylvania, New Jersey, and other Middle and Southern States.

Aside from a number of shipments of from ten to fifteen per cent. pyritous ore from Newfoundland to England, and a few hundred tons of copper produced at Strafford, Vermont, the only important contribution that this group furnished to the world's metal market for the year 1884 was 2,260,000 pounds from the Canadian group of pyrites beds. A few tons of this metal from the Maine mines, and an equally insignificant amount from certain unimportant private enterprises, appear to complete the record for group No. I.

In 1885, the Canadian product was slightly increased, while the Vermont and New Hampshire mines greatly diminished their output. Up to 1890, the production of Canada has not been materially increased.

II. THE LAKE SUPERIOR DEPOSITS.

These have excited such universal interest, from their unique character and their great commercial importance, that there is an abundance of correct and detailed information concerning them in the pages of the *Engineering and Mining Journal*, the *Reports of the United States Geological Survey*, the *Transactions of the American Institute of Mining Engineers*, and in various other publications easily accessible to the student.

The grade of the ore furnished by these extensive conglomerate deposits is very low, ranging from three-quarters of one per cent. of copper to about one and three-fourths per cent., save in the case of the Calumet & Hecla, whose extraordinary extent and richness—four and one-half per cent.—make it a remarkable exception. The figures just mentioned are so low in comparison with the percentage of many less profitable sulphuret mines that it is necessary to explain to those not familiar with the subject, that the occurrence of the copper in the Lake mines in a native or metallic state permits the substitution of an inexpensive mechanical concentration for the costly series of calcinations and fusions necessary in the treatment of sulphide ores. In the case of the Lake ores, a single crushing and washing removes the gangue rock almost completely, leaving the metal in such a condition that a single refining process fits it for use, and yields, in the absence of sulphur, arsenic, or other impurities, a copper of the best quality, which commands the highest price in the market.

The production of this group of mines is very large, and shows a steady increase. For the year ended December 31, 1884, it aggregated 69,353,232 pounds of pure copper, which in 1885 was increased to 72,148,172 pounds, while the striking of the Calumet & Hecla vein at a great depth by the Tamarack Company promises an important addition in the future to these already large figures. The Lake production in 1889 was 86,139,000 pounds, and in 1890, 99,570,000 pounds.

Regarding these mines from an economic standpoint, we find that the veins producing the great masses of native copper, for which this district is so celebrated, have not proved so persistent nor so uniformly profitable, as the veins in which the copper is deposited in more minute particles.

A totally different series of mines were those that were started in the porphyritic diabase, that is an important and persistent feature of the geology of this part of the country.

These so-called "Ashbeds" carry more or less copper throughout a very great extent, but are only rich enough to pay for working within a small area.

The Kearsarge, Wolverine, Pewabic, Franklin, and Atlantic are, or have been, noted producers of copper from these ashbeds. But the best known of the entire lot is the Quincy mine, which differs greatly from most of the other amygdaloid beds in carrying nearly half its copper in comparatively coarse fragments, though in nowise resembling the "masses" of the group immediately preceding. The average yield of the Quincy ore is 2 per cent., and as this mine has already reached a depth of about 4,000 feet on an incline, it shows how little weight need be attached to the fear of greatly increased costs as depth is gained. Of course, if there is much water to be pumped to the surface, the condition becomes different.

The celebrated Atlantic is also an amygdaloid mine, and is situated very near the Quincy. It has a world-wide reputation of having brought the costs of mining and milling down to the lowest conceivable point of economy. Its yield is extremely uniform, usually averaging 0·75 per cent. Last year it dropped to 0·6 per cent., which is only 12 pounds copper to the ton of ore.

This, at 15 cents per pound, would represent a gross cash yield per ton of $1.80. Out of this small amount must come the cost of mining from a depth of 1,600 feet; of constant and very extensive development of ore; of crushing through a comparatively fine screen, and concentrating the copper into a 70 or 80 per cent. product fit for the smelter; of freighting this product to the smelter, and paying a comparatively high price for the single furnace operation, which is a combined smelting and refining; of transporting the ingot copper to a market at a great distance, of selling the copper, and of the general office and managing expenses. And last of all, though by no means least important, of paying the dividends to its stockholders.

The Atlantic mine and its management present an excellent subject for contemplation by the owners of many Western copper and silver properties. Not that its methods can be always followed, or that its own manager and foremen, confronted with the difficulties of mines and ores that are unfamiliar to them, make a better showing than other men, as many unfortunate investors that have pinned their faith on Lake Superior mining captains, have learned. But the strict economy and system here practiced, and indeed rendered necessary by the struggle for existence, may be copied with immense advantage by the majority of the Western.

The mines that at the present time are the great copper producers of the Lake region are those that are following out one of the several well-known but otherwise comparatively barren, conglomerate beds.

The first corporation to demonstrate that these beds were, in places, commercially valuable, was the well-known Albany and Boston Mining Company, in 1863, I believe.

Its success was only of short duration, but was soon followed by the organization of the celebrated Calumet & Hecla Company, in 1865, one of the most profitable mining ventures of modern times in proportion to the money paid in at the start.

The Allouez, Osceola, Tamarack, and Tamarack Junior are the other principal conglomerate mines, though the Osceola for several years has transferred its attention to an amygdaloid bed that crosses its territory.

The Calumet & Hecla had the extraordinary good fortune to include within its original boundary lines the largest and most important portion of the only really profitable conglomerate bed ever known in the Lake region. Although this bed extends for a great length, and is a prominent feature in the geological landmarks of the district, it is only for a length of about three miles along its strike that it is profitable. The Calumet & Hecla happened by great good fortune to sink its first shaft almost exactly in the center of this magnificent and almost unparalleled ore-chute. Some seventeen shafts have been sunk along the outcrop of this vein, and an extreme depth of 3,800 feet has been reached on an incline. This incline, or, in miners' parlance, "dip," being 37½ degrees, the greatest perpendicular depth is about 2,310 feet. This point is between the 29th and 30th level, and as, in spite of this great depth, there is no excessive heat, we can see no reason why depths may not be reached in this mine to far exceed anything yet attempted in subterranean work. Of course it will depend mainly upon the extent in depth of the ore-chute, which has never yet shown the slightest indication of either weakness or poverty.

Many mining men do not understand how it is possible for the Tamarack Company to have tapped the Calumet & Hecla vein below its [then] deepest workings, by means of a perpendicular shaft, when United States mining laws allow each owner to follow his vein in depth indefinitely, following all its dips and angles, and regardless of whose surface ground it may run under.

And, indeed, this remarkable enterprise in prospecting would not have been possible had the State of Michigan been under the control of the United States Mining Code.

But most of the older States have their own mining regulations, and Michigan, in common with many others, has decreed that mining lands shall be bounded as any other lands; that is, by vertical lines, or rather, by lines running toward the center of the earth. Therefore, if a vein pitches at all from the perpendicular, it is evident that in time it will pitch outside of the boundary lines of any tract, however large, and the extreme regularity of pitch and value in the Calumet & Hecla bed tempted the Tamarack people, who owned a tract into which this bed must penetrate at a certain depth, to sink a vertical shaft to cut the bed at this great depth. The shaft was sunk with great rapidity and skill to a depth of 2,275 feet, at which point it cut the Calumet conglomerate bed in all its richness, as had been anticipated, and ever since, the Tamarack has been a large and profitable producer.

But it must not be understood that the Calumet & Hecla is throughout its entire course cut off by the Tamarack at this comparatively slight depth. If so, its career would already be nearly run, and it could not possess the deep workings above described. It is only a comparatively small tract of the Tamarack Company that thus thrusts itself into the Calumet grounds, though further explorations have opened up enough territory to insure a long and profitable future of the newer enterprise, as well as the older.

The occurrence of native copper at the Ray mine in the little Canyon of Mineral Creek, which empties into the Gila River just below the town of Riverside, Pinal County, Arizona Territory, attracted considerable local interest at one time.

But, mineralogically speaking, it possesses but little interest, as the native copper results simply from the decomposition of sulphides or similar compounds.

The mine consists of a strong dike of diorite, which on cooling has become so cracked and fissured, that on its subsequent permeation by watery solutions carrying copper, this metal has been deposited in a thin film on every surface of these innumerable cracks and planes of fissure. This saturation with mineral salts has also decomposed the diorite to a high degree, changing it into a softish material, resembling steatite. It is only in certain limited areas that the metallic copper becomes at all a marked feature, but in one certain winze, between the 50 and 100-foot level, it presents a most tempting appearance, the sheets of native copper being so thick and numerous that it is actually difficult to mine the rock, and, after much difficulty, I have taken out pieces myself, weighing fifty pounds or more, that were simply a framework of native copper, filled in with diorite, and certainly containing 30 per cent. or more of that metal. Unfortunately these rich chutes are very limited, in depth as well as horizontally, the main value of the mine depending upon its immense masses of low-grade ore which, near the top, consists of carbonates and oxides, but within a hundred feet from the surface, changes into copper-glance, which fills the minute fissures of the diorite in the manner already described. A more interesting mine for the geologist or student of vein formation can hardly be imagined. It is mentioned in connection with the Lake mines solely on account of its native copper.

While speaking of the distribution of copper in a metallic form, it seems best also to include the native copper deposits of Santa Rita, New Mexico, which differ so radically from other members of the group in which they geographically belong, that they must be regarded as unique.

In the southwestern corner of New Mexico, a large but ill-defined tract of land is overlain by porphyry, which, although apparently homogeneous, is in reality of several varieties, only one of which is metal-bearing. It is only where this particular formation comes to the surface that the rock is found heavily stained with the salts of copper; and on being followed beyond the limit of destructive atmospheric influences, it carries a fair but very variable percentage of copper in sheets, nodules, threads, etc.

The most striking difference between these deposits and those of Lake Superior is in the degree of decomposition of their metallic contents. In the Santa Rita deposits, this decomposition has progressed to such an extent as to have transformed the entire nodule of metal into an oxide or carbonate, the red oxide—*cuprite*—greatly predominating, while the two carbonates occur chiefly as stains and films. Even such pieces of metal as seem to have escaped oxidation, and to have retained their original form and appearance, are found, on close examination, to consist of numerous thin plates of metal, separated by a layer of oxide, while the entire mass is so thoroughly decomposed that little difficulty is experienced in grinding the greater proportion of it into a powder.

This condition of the ore naturally produces a most unfortunate complication in the subsequent process of mechanical concentration, and leads to enormous losses, especially when stamps are used for crushing the ore.

III. THE MOUNTAIN SYSTEM OF VEINS.

In this group are all the deposits of copper occurring in the Rocky Mountains and Sierra Nevadas north of the great carbonate districts of Arizona and New Mexico.

It is an uncertain division, geographically speaking, and unsatisfactory geologically, as it contains a heterogeneous collection of mines and minerals, scattered over an immense and ill-defined tract of country. An enumeration and very brief notice of the few great centers of commercial importance will also include a sufficient diversity of ores to serve as types for the whole.

Most northerly, and by far the most important of all, are the deposits at Butte City, Montana.* The copper minerals occur here in a granite formation, in some places approaching gneiss in structure and appearance. The veins are undoubtedly true fissures, and constitute two groups of different geological periods, crossing each other at an angle of about 60 degrees, although no exploration has been done at the points of intersection. The east and west veins have been explored to a much greater extent than those of the other group, and are apparently of much greater value. They are of unusual width, from 5 to 40 feet, perhaps averaging 7 feet, and vary greatly in pitch, although usually approaching the vertical. They seem to stand in some constant relation to an accompanying band of a decomposed material resembling porphyry, varying in width from 100 to 1,000 feet, and near the longitudinal axis of which they are situated.

The east and west veins are alone worthy of particular notice in this place. They vary among themselves in the minerals that constitute their value. The Anaconda and neighboring properties carry in the decomposed zone chiefly a copper glance of greater or less purity, while the Parrot portion of the great vein, with its satellites, yields principally an ore resembling bornite (peacock, or horseflesh ore), which varies greatly in its chemical composition. A striking feature of these veins is, that they carry little or no copper for a space of from 50 to 500 feet (average about 250 feet) from the surface down to water-level, when the rich minerals already described appear suddenly, changing from poverty to wealth in the space of a few feet, instead of gradually, as is usually the case in other districts.

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* For a detailed description of the mines and metallurgical works of this district, see a paper by the author, entitled "The Mines and Reduction-Works of Butte City, Montana." *United States Geological Survey, Mineral Resources*, Albert Williams, Jr., 1885.

This rich zone has evidently derived its principal value from the leaching out of the surface portion of the vein, and its unusual extent may be inferred from the fact that many of the workings have penetrated it for 300 feet or more, without finding any serious diminution in its copper contents; but some of the large mines have already worked through it into the unaltered ore of corresponding low grade.*

The average value of the second-class Butte ore, as extracted, is about 10 per cent., and perhaps one ton in ten is set aside as first-class, averaging 30 per cent. The second-class ore is well suited to mechanical concentration, the principal drawback being the accompanying pyrite, which, though valuable as a flux, prevents the production of so rich a concentrate as economy would dictate, and adds materially to the expense of calcination. An important commercial advantage is enjoyed by the owners of the Butte mines in the silver that occurs in amounts varying from one to four cents for each pound of copper. Some 18,000 tons of such ore, smelted under the charge of the author, yielded, according to the assays on which the product was sold, 0·5757 ounce of silver to each per cent. of copper, or $3\frac{1}{6}$ cents silver, at $1.10 per ounce, to each pound of copper.

As the silver was not leached out with the copper from the upper portions of the vein, it follows naturally that the re-deposition in the rich zone of this vast quantity of non-argentiferous copper, would greatly lessen the ratio of the more valuable metal to the copper. Further, that if this reasoning were correct, we should expect the unaltered ores in depth, and that were not diluted with extraneous copper, to be proportionately richer in silver than the ores from the rich copper-zone. And this is found to be the fact as far as the unaltered zone has been penetrated.

In order to give some idea of the percentage of these ores at the present time, I quote from the reports of the Boston and Montana Company, published for the year 1890. The actual yield of all the ore mined and treated by this corporation for that year was 9·36 per cent., and the copper produced aggregated 26,942,298 pounds. But this is a very high average, a large proportion of the yield having been derived from the rich, altered ores.

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* The writer desires to acknowledge the assistance of Mr. James E. Mills in preparing his brief statement of the geology of the Butte copper mines.

The yield of the Anaconda and affiliated mines, as obtained by Mr. James Douglas by comparing the amount of ore reported by this company to have been transported to its works with its copper production, was 5·9 per cent.; and as considerable rich, altered ore was worked during the year, it is evident that, in the near future, all the Butte mines must prepare to handle an ore that does not average above 5 per cent. in copper, and possibly 5 ounces in silver. Whether this can be done at the present rate of wages, fuel, and transportation, remains to be seen.

An almost exactly analogous occurrence of these rich purple ores gave originally that great impetus to mining that has placed Chili, up to a recent date, at the head of the copper-producing countries of the world, and a steady, though by no means rapid, improvement in practice and apparatus has enabled her to maintain her output, even though the rich, altered ores have been long since exhausted, and have given place to the 7 or 8 per cent. pyritous material that forms the normal filling of these and most other copper veins. The complete disappearance of the more valuable mineral occurs at a depth of about 500 feet in the deepest Chilian mine on record, the Piqué mine.

The product of the Butte district for the year 1884 is closely estimated at 41,500,000 pounds; for 1885, 67,798,864 pounds, while in 1889 it was increased to 104,200,000 pounds, rising for 1890 to 122,950,000 pounds.

The remainder of No. III. group is of a miscellaneous character, geologically, mineralogically, and geographically.

The next most noteworthy occurrence of copper is in the fissure-veins of Gilpin County, Colorado, which traverse a granite formation, and are principally important for their value in the precious metals. The copper contents rarely exceed 1 per cent. of all the ore extracted, or 5 per cent. when only the first-class or smelting ore is considered. The gangue is invariably quartz or decomposed feldspar, and the metal occurs almost exclusively as chalcopyrite, in company with small quantities of zinc-blende, galena, various antimonial and arsenical compounds, and a much larger amount of pyrite.

The new San Juan region promises to add a considerable amount of copper to Colorado's quota. The metal occurs there principally as a constituent of argentiferous tetrahedrite. The statements of ore-buyers, verified by personal examination, show an average tenor of about 3½ per cent. of copper for the ordinary ore as shipped from these mines. Their silver value varies from 20 to 500—average about 60—ounces to the ton. A limited amount of copper is furnished from other districts of Colorado, but is in no case mined in sufficient quantities to justify an independent business apart from the precious metals, which almost invariably constitute 90 per cent. or more of their value, as expressed by the price received for the ore at smelting-works. The product, therefore, is insignificant, and, when added to that of the districts already noticed, will not raise the total product of group No. III. for 1884 to much above 44,000,000 pounds. In 1890, Colorado produced 6,760,000 pounds.

IV. THE SOUTHERN CARBONATE DEPOSITS.

It is to this group of oxidized ores that the attention and capital of business men were principally directed a few years ago, and although the deposits of this nature are almost limitless in number, and the labor and expense of producing metallic copper from minerals that have already been prepared by nature for the simple fusion that is alone necessary, are comparatively slight, the high expectations formed have seldom been realized. The unfavorable character of the country, the scarcity of fuel and water, the expense of transportation, the distance from central authority, and, above all, the eccentric and uncertain character of the deposits, have brought about this result, and the copper market has been overloaded, and many valuable deposits exhausted, without any corresponding advantages to the promoters.

The most important mines of this division are all situated in Arizona or New Mexico, and differ too much in their characteristics to permit of any general description.

In one class of these mines, notably the Copper Queen, the copper occurs as carbonates and oxides, associated with oxide of iron and ferruginous clays, filling immense caves in the limestone.

In another class of mines, such as those at Clifton, Arizona, the bodies of oxidized ores are irregularly distributed through beds of diorite, the occurrence of the ore beds being apparently determined by intercalated masses of limestone, which have played an important part in either the deposition or alteration of the copper, or in both processes. The ore-bodies, although they occur within certain limits, were irregularly distributed, and are of very variable size, and the alteration has not occurred to any great depth.

Although this class of deposits furnishes a certain amount of the very richest ore known to the mineralogist, in the shape of streaks, bunches, and even considerable aggregations of red oxide and the two carbonates of copper, the average percentage of this metal contained in the furnace charges of the most extensive and profitable smelting establishments belonging to this group is not high. A constant yield of 10 per cent. is considered very good, and the dividends afforded by certain of these properties, laboring under the disadvantages of expensive fuel, transportation, etc., result principally from the exceedingly simple nature of the process employed and the remarkably favorable composition of the accompanying gangue rock. This consists usually of a mixture of oxides of iron and manganese, with calc-spar, and for the most part contains just about the proper amount of silica to form, with the constituents just mentioned. an easily fusible slag, and one reasonably free from copper, considering the unusual practice of producing metallic copper and a slag to be thrown away at the first fusion. In cases where the contents in silica exceed the proper amount, very basic ores, containing an excess of the oxides of iron and manganese, can almost always be procured in the immediate neighborhood; or, in default of this, beds of quite pure iron ore and quarries of limestone almost perfectly free from silica are nearly always close at hand. The great purity of the metal produced is also a highly favorable factor in estimating the relative advantages of this group; for, aside from bringing a price nearly equal to Lake copper, it is always sure of a ready sale.

The product of this group, including the small amount referred to in the succeeding paragraph, as coming from unclassified sources, may be safely estimated for the year 1884 at 24,000,000 pounds, dropping in 1885 to 22,706,000 pounds, and rising in 1889 to 31,600,000 pounds, and in 1890 to 35,720,000 pounds.

A considerable number of mines and deposits that cannot be consistently brought under any of the four divisions enumerated still exist; but aside from certain sulphureted veins in Nevada County, California, which are interesting chiefly on account of the method employed for the beneficiation of the ore—leaching and precipitation in revolving barrels—and the Walker River mines in Esmeralda County, Nevada, but few deposits are worthy of note. The Nacimiento copper quarries in Central New Mexico,* the Oscura copper-fields in the Oscura Mountains, New Mexico,† and the Great Belt copper deposits in Texas present certain curious and interesting features to both mineralogist and paleontologist. The metal in each place occurs in the shape of petrifacts of shells, fishes, and palm-leaves, branches, and twigs—all changed completely into an impure variety of copper glance, and found in that same Permian formation that at Mansfeld, Germany, and in the Russian Empire, has been, and still is, so prolific in copper.

Since the preceding paragraph was written, there has been little change in the Arizona mines, and but a slight increase in their output.

In describing the principal mines, I shall quote from a paper by the President of the Copper Queen mine, than whom there is no better authority on the subject.‡

There are only three groups of mines in Arizona now producing copper in any considerable amount. These are the districts of Clifton, Bisbee, and Globe.

The Clifton district, in the southeastern corner of the Territory, so far as production is concerned, is divided between a Scotch organization, known as the Arizona Copper Company, and the Detroit Copper Company.

At Globe, in Pinal County, the Old Dominion Company, on the old Globe mine, is the only important producer, while at Bisbee, in the extreme south of the Territory, the Copper Queen

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* See pamphlet by F. M. F. Cazin, for a full and accurate description of these mines, and estimate of their value.

† See a paper by the author in the *Engineering and Mining Journal* of November 18, 1882, for report on these properties and the surrounding country.

‡ The "Copper Resources of the United States," by James Douglas.

Company, under its new management, has absorbed by purchase most of the surrounding promising claims, though without ever finding anything to compare with its original famous deposit, which is still largely productive.

As experience and knowledge have been gained by exploration and study, it is found that there is a much closer analogy between these different deposits than was at first supposed. Oxidized ores of copper are found, to a greater or less extent, throughout almost all of the peculiar, abrupt mountain ranges, so characteristic of Arizona. And wherever they are found in any quantity, an examination of the adjacent rocks is pretty certain to reveal limestone [probably of the Carboniferous Age, and of a thickness varying from a few hundred to 3,000 feet], quartzite, and dikes of some volcanic rock, usually of a porphyritic texture.

The copper ore itself, as Mr. Douglas points out, occurs almost invariably in, or very near, this carboniferous limestone, which has been mainly influential in effecting the oxidation of the ore and the removal of the sulphur with which it was no doubt originally combined. This process of oxidation in many instances extends to a great depth, for the decomposition of the ore itself has furnished solvents that act most powerfully upon the limestone, especially at the edges of a shrinkage-crack, or wherever they can obtain a point of attack, and on this account, water has penetrated the limestone and decomposed its ores to a far greater depth than the ores found in the adjacent feldspathic rocks.

In Bisbee the ore is confined exclusively to the limestone, occurring often in chambers of vast size, and which at first were supposed to have been caves in the limestone filled with these massive deposits of oxidized ores in place, but now are thought to have resulted from the gradual decay and solution of the limestone, and the interstitial exchange and deposition of an atom of the copper mineral in a sulphide form, and which has since become oxidized.

In Clifton and Globe, the ore is not so strictly confined to the limestone, but is found mainly in the contact between the limestone and granite or sandstone.

A peculiarity of all these mines is the occurrence of sometimes extensive bodies of unaltered sulphide ores, often in the higher levels, and surrounded on all sides and below with thoroughly oxidized ores. It is difficult to see what causes have been at work to prevent their oxidation amidst such universal decay.

The yield of the Arizona mines is difficult to get at, as so much sorting is done, and so much depends on the cost of freight, that only ore above a certain percentage can be profitably delivered to the furnace.

At the Copper Queen, the average yield of the ore smelted is 8 per cent., but Mr. Douglas tells us that to obtain an ore of this grade, nearly an equivalent amount of lean ore is stowed away in the stopes.

The yield at Clifton and Globe is necessarily considerably higher, owing to the 100-mile wagon-freighting of the coke to Globe, and the fact that the Clifton ores are so siliceous that much barren limestone has to be added to the furnace charge.

The Copper Queen ores, consisting of oxidized copper-compounds, with a gangue of ferric oxide and some alumina and silica, are probably as favorable smelting ores as the world has ever seen.

The Verde mine, in the northern part of Arizona, has been at times an important producer, and is still considered a most valuable property. Much of its value consists in the precious metals, which are, however, distributed very irregularly throughout the copper ores.

The ores of this mine in all respects are different, both in occurrence and composition, from the Southern carbonate mines. They consist of lens-shaped masses in crystalline schists, and are oxidized only on the surface, the carbonates being soon replaced by the rich altered sulphurets, similar to the Butte ores.

THE FUTURE OF COPPER IN THE UNITED STATES.*

A careful study of the history and present condition of the copper-producing district of the United States, as well as of the tracts of country still partly or entirely virgin, where lean, pyritous ores of copper are known to exist, seems to point to the following conclusions:

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* This chapter was already completed when I first saw the paper of Mr. James Douglas already referred to, on "The Copper Resources of the United States." My own conclusions so closely resembled his [which, to be sure, is not remarkable, as we both reasoned from the same premises], and the form in which they were expressed so closely followed his paper, that I have thought it best to supplement my conclusions by some new ideas that I found in his, and credit the entire paragraph to Mr. Douglas.

Preserving the geographical distribution of copper that has already been adopted in the preceding paragraphs, we have first to consider what may be expected from the Atlantic Coast beds.

At present the only copper produced in this district of any importance is from the Ely mine, in Vermont, which has awakened from its lethargy, and under more economical and sensible management than it has ever before enjoyed, is producing now at the rate of something like 1,500,000 pounds annually. This all comes from the bottom of the shaft, at a depth of nearly 3,000 feet on an incline, or about 900 feet perpendicularly from the surface. And it is with great satisfaction I can report, from late personal observation, that there is not the slightest sign of any falling off in either the quantity or quality of the ore at this great depth.

When one reflects that this same great bed strikes through the country for many miles, cropping out at intervals, and everywhere showing strong bodies of pyrrhotite, with more or less chalcopyrite intermixed, it is surprising that so little exploration has been accorded to it, especially as the copper is of extra good quality, fully equal to the best Arizona copper.

The Maine pyrites mines are practically idle, their grade having been too low to repay the cost of working. But a slight rise in the price of copper would enable several of these mines to exist, as well as stimulate the search for others.

The noted Ore Knob mine in North Carolina has probably seen its best days, but vast quantities of low-grade ores are said to still exist in the Pyrrhotite mines of Ducktown, Tennessee, and it is probable that these will some day again become producers.

In Georgia and Alabama, there are considerable bodies of pyrites suitable for acid-making that will in time, no doubt, add their mite to the general production of copper, but it is an unfortunate fact, that there is as yet no sign in any part of our country of those immense cupriferous masses of iron bisulphide that have rendered the Iberian peninsula so famous, and added so greatly to the wealth of England.

The copper-bearing slates and sandstones of Pennsylvania and New Jersey, and the fissure veins of Maryland, are but small affairs, and unless greatly augmented by future discoveries, cannot be considered as of the slightest importance in the future of the metal under consideration.

Unless, therefore, some radically new, and as yet unsuspected, source of copper-production is discovered, a most unlikely contingency, we cannot look to the Atlantic Coast for any considerable supply of copper in the future.

If this district possessed gigantic veins or beds of iron pyrites, even quite low in copper, we might expect in time a large production of the more valuable metal from this source, as the total production of our sulphuric acid therefrom would only be a question of time.

But they apparently do not exist, and even to-day it is found cheaper to import cupriferous pyrites for acid-making from the Spanish mines than to use our own less favorable and non-cupriferous material, which also has to stand railroad transportation instead of the far cheaper ocean-freight.

Nor do the carbonate deposits of the Southwest promise to be any factor of increasing importance to our copper industries.

Not a single new mine of importance has been opened since the days of the old mines already described, and to which Arizona's fame is due. Not but that there are many scattered deposits of local importance throughout her vast extent of copper-bearing rocks that, in the aggregate, will produce considerable copper for an indefinite period. But no one district can be selected that is at all likely to surpass those now in operation, and they themselves are already straining their resources, and feeling the need of every possible advantage accorded to them by the cheaper freights and fuel of the present day, to offset the lower grade of their ores, the increased expenses of deep mining, and the extensive exploratory work essential to their existence. To these expenses, the cost of pumping on a large scale will, no doubt, be added when a certain depth is reached, and Arizona can hardly be counted on to materially increase her output in the future, although she may hold her own for many years, even this fact depending mainly upon the ruling price of copper.

Although the ores of the Copper Queen have remained quite thoroughly oxidized to an unusual depth, and still show no signs of changing, this is by no means the usual rule of this Territory, and most of the mines, before reaching a great depth, will lose the enormous advantage that they now mostly possess, of having their calcining done by Nature, and being able to produce copper by a single fusion of the simplest character.

What has just been said of the Arizona mines applies in a general way, and on a much larger scale, to the more important Montana mines; the cream of the Butte district has already been skimmed, and the less valuable portions only remain for the future.

This lowering of value will be partly offset by greater skill and economy in mining, and especially in treating, the ores in the future, and the larger companies, notably the Anaconda, with its affiliated Chambers Syndicate, as well as the Butte and Boston and Boston and Montana companies, have wisely obtained possession of large tracts of ground that is still virgin, and to a greater or less extent on the strike of the great ore-course that gives Butte its value.

The enormous plant of the Anaconda Company, when it gets fairly into the valuable ground that has for the past few years been inaccessible on account of the fire, together with the new and extensive plant of the Boston and Montana Company at Great Falls, and the tendency of all the existing works to increase their capacity, will probably, for a time, augment the output of copper from this district, almost regardless of the market price of this metal. For, if the price is low, the companies have to increase their output to pay any profit at all; while if the price is high, they naturally strain every nerve to take advantage of the favorable circumstances.

But the end is certain. Like the great carbonate mines of Leadville, they will soon exhaust their rich, decomposed ores, and have to fall back on their lean, unaltered sulphides, handicapped at the same time with the increasing expenses of deep mining.

But, like Leadville, they will learn to grapple with the difficulties of the situation, and it is quite probable that, with lower wages, cheaper freights, and improved and economically managed plants, they will derive more profit from a 6 per cent. ore at the thousand-foot level than from one of double that percentage at the surface.

It is not probable that any man who saw the rise of Butte City will live to witness its fall as an important and remunerative copper-producer.

As regards Colorado and the rest of the great Mountain System, in which I have included all of the far West, except the strictly local Arizona carbonate deposits, there can be nothing said with any certainty, or even probability, excepting that we have no reason to look for any great increase of copper production. Colorado holds its own, its copper occurring principally as a by-product of the gold and silver ores, for which this State is so famous.

We frequently hear prophecies of the discovery of some new copper-district in the heart of the great Mountain System of the Rockies or Sierras, before which Butte and Arizona will retire abashed, and even the steady flame of Lake Superior's production will wane.

But, as I had written a year ago, and am happy to see substantiated by such an authority as Mr. Douglas, such a discovery is in the highest degree improbable.

Of all the metals, copper is the most obtrusive and the most bold in betraying its presence. A few ounces of its decomposed compounds will stain a whole mountain-side green or blue, and a very small quantity will so affect the waters of all streams running over it, especially in the dry season, that the attention of the prospector is immediately attracted to the fact, and he is sure to find out the cause.

There are no such hidden recesses in this country as are popularly supposed to conceal these wonderful deposits. Possibly within the heart of the lava beds in Oregon, or in some of the remoter and more inaccessible depths of the Dakota "Bad Lands," there may be a few square miles where the foot of prospector has not yet penetrated. But that is only because these districts are well known to contain no valuable minerals.

Wherever there is the least chance of the occurrence of ores, and ten thousand wild places where there is not, the prospector has penetrated over and over again, until every nook and corner of our mountains is as familiar to some one or more mining men, as are the streets of a city to one who is born there.

It has been my lot to spend many months during the past twenty years in the mountainous districts of new mining regions; but I cannot recollect that I ever yet wished to visit any valley, no matter how remote, nor any mountain peak, however inaccessible, that I could not have found a guide, who knew every foot of the ground, and had broken, with his prospecting pick, samples from almost every promising ledge or formation.

There are many districts, especially within the limits of Utah, California, and Nevada, that to-day offer flattering inducements for examination, and no doubt, in the aggregate, will produce many million pounds of copper within the next decade, but there is no reason to expect any overwhelming flood of this metal from any new mines, such as was poured upon the world's market from the almost simultaneously-discovered surface ores of Montana and Arizona, and the cupriferous pyrites of the Iberian peninsula.

The Lake Superior mines seem to offer the greatest probability of a future important increase in copper-production, and this is fortunate indeed, considering the enormous expansion of electrical interests, and the value of this especial copper for that purpose.

At the first glance, this fact may not be so obvious, especially when we see the great depth already reached by the only important mines in that section, and the knowledge we possess of the limitation longitudinally of the ore-chutes on which they depend for their entire product.

But, as Mr. Douglas points out, the Keweenaw rocks extend beyond Michigan and Wisconsin, way through into Minnesota, where they are still copper-bearing, though, so far as yet known, to a very limited extent. But their out-crop is so heavily covered through much of their course by swamp and alluvium, that but little is known of their real value, and it is rather probable than otherwise, that where the theater of action remains the same, as evinced by the identity of the geological formation, the peculiar conditions that have produced a Calumet & Hecla or a Quincy, may be repeated many times.

And it is well known that in Michigan itself, there is an almost boundless area of low-grade conglomerates and amygdaloids carrying one-half per cent. of copper or thereabouts, and thus approaching the standard that the Atlantic mine actually finds profitable. So that a slight increase in the price of copper would probably render profitable a tract of country extensive enough to supply the world for a long period.

Therefore there need be no fear of a dearth of copper at reasonable prices for more than one generation to come, while, on the other hand, there is little reason to expect any sudden important increase in the production of this most important metal.

CHAPTER III.

METHODS OF COPPER ASSAYING.

THE first step usually taken in the treatment of an ore of copper is to learn its value by determining the proportion of that metal that it contains. This process is called assaying, as distinguished from chemical analysis, which includes the further investigation as to the general composition of the ore.

We shall confine our discussion in this place to assaying only. The assaying of any given parcel of ore is necessarily preceded by the process of *sampling*, by which we seek to obtain, within the compass of a few ounces, a correct representative of the entire quantity of ore, which may vary in amount from a few pounds to several thousand tons. As a rule, it will lessen the chance of serious error in very large transactions, to divide the lot into parcels of not over fifty tons each, and sample each of these lots by itself.

The utmost care and vigilance in sampling and assaying should be required at every smelting-works, both in the interest of the works and in that of the ore-seller.

Until quite recently, it has been customary to sample lots of ore by quartering them down, rejecting a certain proportional part at each successive operation, and reducing the size of the ore fragments as the quantity to operate on diminishes. This is a laborious and expensive method, and in the case of finely pulverized ores, may well be replaced by the use of the "split shovel," or one of the many automatic sampling machines that have been invented.

But since the establishment of public sampling works at most of our great mining centers, where the correctness of the sample is guaranteed by the works, which distribute packages of each lot of ore to the agents of the various rival smelting companies, for them to assay and bid upon, the vast quantities of ores handled, and the importance in many instances of retaining the lump form of the ore, as essential to the subsequent metallurgical operations, have imperatively demanded some method of automatic sampling that shall be rapid, accurate, and equally applicable to ores in both the pulverized and lump form.

The means hitherto employed all depend upon the same general principle of cutting or dividing a falling stream of ore by means of flanges, fingers, or traveling buckets, in such a manner as to obtain a certain desired proportion of it for a sample.

While many of these devices work admirably upon pulverized ore, free from dampness or foreign obstructing substances, they are apt to give entirely unsatisfactory results upon a mixture of fine and coarse ores, while the presence of strings, chips, rags, etc., usually clogs them and deranges their working.

Mr. D. W. Brunton, of Denver, Colorado, whose paper I have freely used, has invented an automatic sampling-machine that is apparently free from all the defects enumerated, and which has been shown by practical trial to be equally applicable to coarse, fine, or mixed ores, while it cannot be clogged by foreign bodies of any reasonable size.*

Brunton overcomes these difficulties by deflecting the entire ore-stream to the right or left, while falling through a vertical or inclined spout. By a simple arrangement of movable pegs, in connection with the driving gear, the proportion of the ore-stream thus deflected into the sample-bin may vary from 10 to 50 per cent.; the latter amount only being required in coarse ores of enormous and very variable richness, while for ordinary lump ores, from 10 to 20 per cent. is the maximum required.

Instead of passing the sample-stream of ore into a bin, this system may be still further perfected by leading it directly to a pair of moderately fine rolls, the product of which is elevated to a second similar sampling-machine, from which the final sample drops into a locked bin.

Six months' constant experience with this sampler has shown that 10 per cent. of 20 per cent., or 2 per cent. of the original ore-parcel, is usually quite sufficient; though in exceptional cases, 15 per cent. of 30 per cent., or 4½ per cent. of the ore, may be required.

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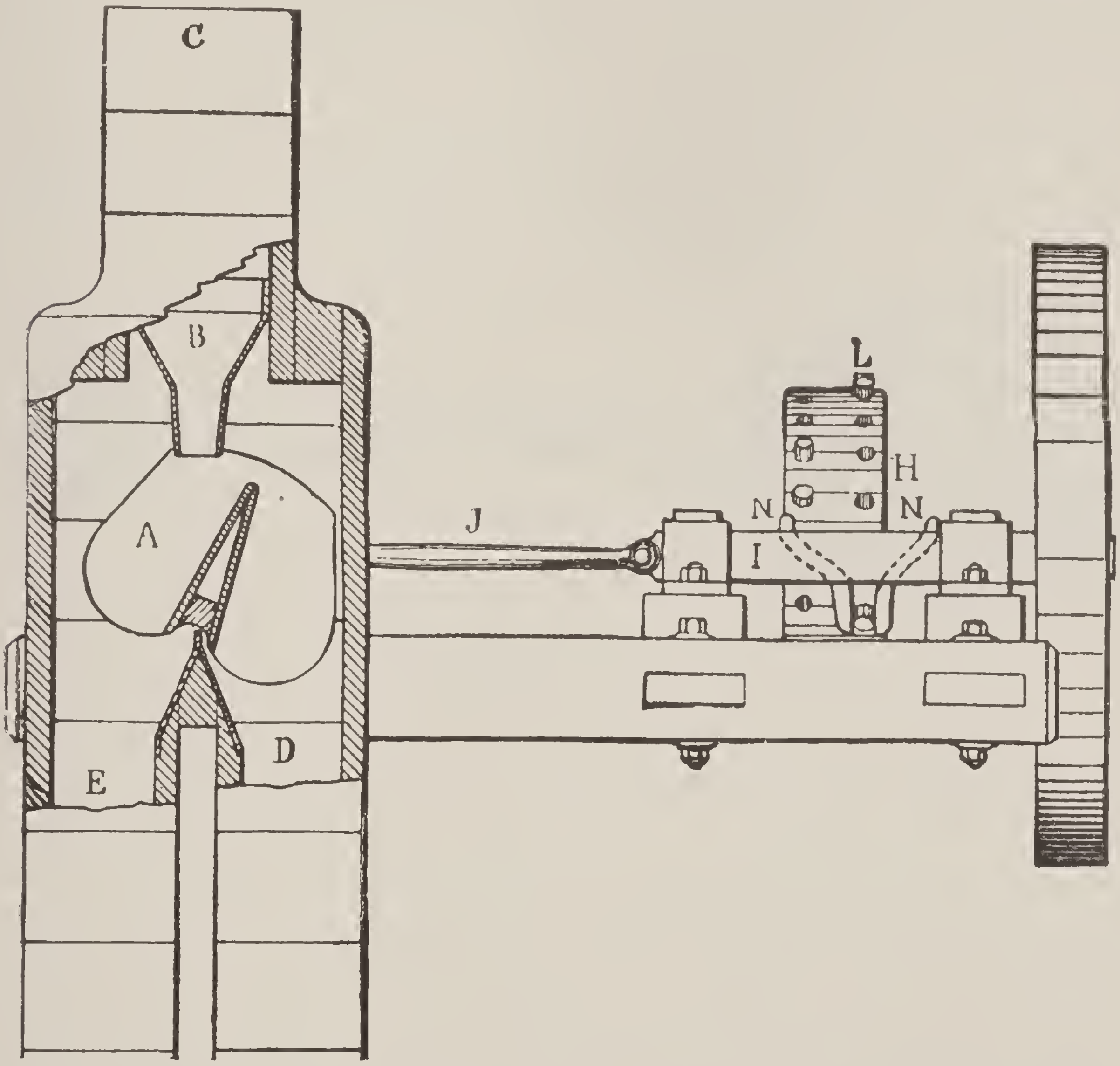
* See *Transactions of the American Institute of Mining Engineers*, Vol. xiii., page 639, for drawings and full description of this sampler.

The two machines are driven at different speeds, to prevent any possible error that might arise from isochronal motion, and by careful tests of this machine in resampling lots of ore, the limit of error has been found less than one-fourth of one per cent.; while even the best hand-sampling may vary two per cent.

The fact that the division is one of time and not of ore is one of the most important features of this valuable invention, as it consequently is forced to deflect the exact proportion of the ore-stream for which it is set; whether coarse or fine, wet or dry, light or heavy. A still more recent invention of Mr. Brunton's, is the quartering shovel, described in the *Engineering and Mining Journal* of June, 1891.



BRUNTON'S SAMPLER.

The determination of the moisture present in any given parcel of ore is also a matter of much importance; and probably more inaccuracies attend this apparently simple process than any other of the preliminary operations.

This determination must, of course, take place as nearly as possible at the same time that the entire ore parcel is weighed, as otherwise the sample may lose or gain moisture.

In lump ores, it is difficult to obtain a correct sample, even for moisture, without some preliminary crushing, and to save labor, it is best to use a portion of the regular assay sample for this purpose; the accurate weighing of the entire ore parcel being postponed until just before or after the sampling, and the portion reserved for the moisture determination being placed in an open tin vessel, contained in a covered metal case, having an inch or two of water on its bottom, in which the sample tins stand.

From one-fourth to one-half pound of the sample is usually weighed out for this determination, and dried under frequent stirring, and at a temperature not exceeding 212 degrees. While it is always important to keep within the limit of temperature just mentioned, it is especially the case with certain substances which oxidize easily. Among these are finely divided sulphides, and above all, the pulverulent copper cements obtained from precipitating copper with metallic iron from a sulphate solution.

BRUNTON'S QUARTERING SHOVEL.

Such a sample, containing actually 5½ per cent. of moisture, showed an increase of weight of some 2 per cent. on being exposed for thirty minutes to a temperature of about 235 degrees Fahr.

Certain samples of ore—especially from the roasting furnace—are quite hygroscopic, and attract water rapidly after drying.

In such cases, the precautions used in analytical work must be employed, and the covered sample weighed rapidly, in an atmosphere kept dry by the use of strong sulphuric acid.

The sampling of the malleable products of smelting, such as blister copper, metallic bottoms, ingots, etc., can only be satisfactorily effected by boring a hole deeply into a certain proportional number of the pieces to be sampled.

Where such work is only exceptional, an ordinary ratchet hand-drill will answer, but in most cases, a half-inch drill run by machinery is employed.

The chips and drillings are still further subdivided by scissors, and as even then it is difficult to obtain an absolutely perfect mixture, it is best to weigh out and dissolve a much larger amount than is usually taken for assay, taking a certain proportion of the thoroughly mixed solution for the final determination.

The various means employed in the laboratory for the determination of the percentage of copper in any substance are given in the standard works so fully and clearly, that a mere enumeration of the four methods that the author deems necessary and sufficient for the assay department of any copper works would probably suffice. But having been at considerable pains in former years to determine the causes and extent of the inaccuracies inseparable from certain of these methods, and also having noticed various essential precautions, not mentioned in the textbooks on this subject, the author has introduced a few original observations where they seem required.

The four methods of assay that are quite sufficient for any commercial or technical laboratory, and yet that are every one essential if it be desired to fulfill every condition to the best advantage, are:

I. Titration with potassium cyanide (KCy).
II. Precipitation with zinc (or iron).
III. Colorimetric determination.
IV. Electrolytic.

To these may be properly added the Lake Superior fire assay, as peculiarly suited to its local conditions. As the Swansea fire assay for copper is described in every English metallurgical work, and as the reasons for its adoption in this country can hardly be imagined, it is omitted.

I. TITRATION WITH POTASSIUM CYANIDE.

This well-known and rapid method, usually called "The Cyanide Assay," depends upon the power possessed by an aqueous solution of potassium cyanide to decolorize an ammoniacal solution of a copper salt, and is, under proper conditions, quite accurate enough for ordinary purposes. These conditions are as follows:

The use of measured and constant amounts of acid and ammonia.

The cooling of the ammoniacal copper solution to nearly the temperature of the surrounding atmosphere before titration.

The intimate mixture of the cyanide solution, as it drops from the burette, with the copper solution, and a sufficient, though accurately limited, time in which to accomplish its bleaching action.

The establishment of a certain fixed shade of pink at the standardizing of the cyanide solution, to which all subsequent assays must be as closely as possible approximated in color for the finishing point. This renders it impossible for any chemist to work with another person's solution until he has first standardized it himself, and determined its strength according to his own custom.

The absence of zinc, arsenic, and antimony, whose presence has long been known to seriously vitiate results, though exactly to what extent has never been demonstrated, until a series of experiments on this point was carried out in 1882 under the direction of the author, and still more recently by Torrey & Eaton.

From a long list of results, some of them even contradictory, the following deductions were drawn:

The presence of zinc in quantities below 4½ per cent. has no perceptible influence on results.

Five per cent. of zinc, in a siliceous ore of copper, containing no other metals except iron, caused a constant error on the plus side of about 0·22 per cent., which increased in a tolerably regular ratio with an increased percentage of zinc.

After eliminating a few results that varied very greatly and unaccountably from all others, an average of about six determinations of each sample yielded the following figures. The ore just described was used in every case, and the zinc added in the shape of a carefully determined sulphide, allowances being also made for the increase in the weight of the ore sample resulting from this addition of foreign matter

Ore free from zinc,				11·16 per cent. copper.	
No. 1 with	4 per cent. zinc,			11·46 “	“
“ 2 “	4½	“	“	11·55 “	“
“ 3 “	5	“	“	11·72 “	“
“ 4 “	6	“	“	12·1 “	“
“ 5 “	8	“	“	13·2 “	“
“ 6 “	10	“	“	13·3 “	“
“ 7 “	15	“	“	13·9 “	“
“ 8 “	20	“	“	13·8 “	“

The presence of arsenic and antimony in much smaller proportions—1 per cent. or less—may cause errors on both plus and minus sides to the extent of one-half a per cent. or more, and in larger quantities will generally render the test totally unreliable.

Another indispensable and oft-neglected precaution is the testing of the precipitate of hydrated oxide of iron caused by the addition of ammonia to the original solution. This bulky precipitate, especially in the case of mattes and highly ferruginous ores, will retain a considerable amount of copper which even the most careful washing will not remove, but which may be speedily determined by dissolving the precipitate in the smallest possible quantity of muriatic acid, saturating with ammonia, and again titrating if any blue coloration is produced. The following results, taken from the note-book of an experienced chemist, who had never been aware of this possible source of error until accidentally mentioned to him by an assayer in the employ of the writer, and who at once instituted careful experiments to ascertain the probable extent of the mistakes that he had made while acting as assayer to large smelting-works, give some idea of the serious discrepancies that may arise from the non-observance of this precaution:

Character of sample.	Without resolution of precipitate.	With resolution of precipitate.
No. 1, pyritous ore	21·2 per cent. copper	23·7 per cent. copper
“ 2, bornite	37·8 “ “	42·4 “ “
“ 3, cupola matte	27·7 “ “	31·2 “ “
“ 4, reverberatory matte	46·4 “ “	47·4 “ “
“ 5, blue metal	57·7 “ “	58·2 “ “
“ 6, white metal	74·7 “ “	75·2 “ “
“ 7, regule	86·2 “ “	86·4 “ “
“ 8, blister copper	97·3 “ “	97·2 “ “

As might be expected, the greatest discrepancies exist in connection with those samples containing the largest amounts of iron, and decrease to nothing as the iron contents diminish.

In the absence of the injurious elements—zinc, arsenic, antimony—the cyanide assay is sufficiently accurate, and, from its simplicity and rapidity of execution, it is peculiarly adapted to the daily working assays from the mine, smelter, and concentrator. In fact, it is the mainstay of the overcrowded metallurgical assayer, and can be used for nearly every purpose, except for the buying and selling of ores and copper products, and for the determination of very minute quantities of copper in slags. It is frequently employed with satisfaction for the last-named purpose, a much larger amount than usual being taken, in order to obtain a solution sufficiently rich in copper to exhibit a reasonable degree of color. Messrs. Torrey & Eaton have published additional investigations of great value on the effect of various substances upon the accuracy of the cyanide method. (See *Engineering and Mining Journal*, May 9th, 1885.)

In their experiments, they employed a cyanide solution capable of showing one-thirtieth of one per cent. of copper, and took every precaution to have all conditions identical during the various tests; all solutions titrated being of the same degree of strength.

Silver and Bismuth.—A solution was made of the following metals:

Copper	·550	gram.
Bismuth	·200	“
Silver	·250	“

The silver was precipitated with hydrochloric acid, and ammonia added after filtering and washing. Two titrations gave:

No. 1	54·90	per cent. copper	
No. 2	54·85	“ “	instead of 55 per cent.

These results show that a solution containing the very unusual proportion of 20 per cent. of bismuth and 25 per cent. of silver can be titrated to within 0·1 per cent. of its value in copper.

Lead.—This metal, being a common element in copper ores and alloys, was introduced into a copper solution in the following proportions:

Copper	·200	gram.
Lead	·800	“

After adding ammonia and allowing the lead precipitate to separate for two or three hours, it was titrated, giving 20·28 per cent. of copper, instead of 20 per cent. Messrs. Torrey & Eaton, therefore, believe that the amount of lead commonly present in ores—from 5 to 40 per cent.—would not injuriously affect the operation.

Arsenic.—Torrey & Eaton titrated, without filtering, a solution containing ·600 gram arsenic, ·400 gram copper, finding 39·8 per cent. instead of 40 per cent. Therefore any ordinary amount of arsenic—from 5 to 15 per cent.—would seem to have no injurious influence.

Ammonia and hydrochloric acid, when indiscriminately used, were found by Messrs. Torrey & Eaton to cause serious errors, the results being influenced to the extent of from ½ to 1 per cent. by any large excess of either.

Lime in large quantity was found to confuse results.

Magnesia had no effect whatever.

II. PRECIPITATION WITH ZINC.

This is simply a modification of the well-known Swedish method, and has been so arranged by Kerl (see his work on assaying) as to be suitable for every variety of ore or product, regardless of impurities. In fact, its chief value in the modern metallurgical laboratory is to take the place of the cyanide method in those cases in which the occurrence of deleterious substances forbids the employment of the latter.

The principal drawback to this method of assay is the delay caused by the precipitation, and the drying and weighing of this precipitate, whose strong hygroscopic quality renders the latter manipulation tedious and frequently inaccurate. This can be easily obviated by dissolving this precipitate, now free from all impurities, and determining the percentage of copper by the ordinary cyanide assay. This modification can be strongly recommended.

III. THE COLORIMETRIC DETERMINATION OF COPPER.

This is reserved almost exclusively for the determination of minute quantities of copper contained in slags, tailings from concentration, and similar products.

Heine's modification of this method, as described by Kerl, is perhaps the most convenient, and with proper solutions for comparison, preserved in bottles of colorless glass and of exactly the same size, yields results that cannot be surpassed. It is seldom employed for substances containing over one and one-half per cent. of copper, and may be relied upon to show differences of $\frac{3}{100}$ of one per cent.; results, however, depending largely upon the skill of the operator, and his capacity for discriminating almost invisible shades of color.

IV. THE ELECTROLYTIC METHOD, OR BATTERY ASSAY.

This is suited to nearly every class of material and every percentage of copper, from the highest to the lowest, and owing to its ease of execution and extreme accuracy, has already largely supplanted the ordinary analytic methods, and bids fair to do so altogether in all important cases. Among those assayers who do not yet practice it, there seems to be an impression that it is difficult of execution, and in several cases under the author's observation it has been abandoned after a few futile efforts. In these instances there must have been some direct violation of the laws governing the generation and transmission of electricity—it being always the battery that was complained of—and as a similar though usually much more extensive and complicated form of battery is under the charge of every telegraph operator, the disappointed assayer should feel encouraged to persist.

Messrs. Torrey & Eaton have also investigated the effect of various substances upon the battery assay, and have arrived at the following results, which are not quite so favorable as the author's experience in practice has been: *

"*Silver*, when present in any considerable proportion—from 1 to 3 per cent.—gives too high a result. It should always be removed by hydrochloric acid.

"*Bismuth*, even when present in small quantity—½ per cent. —is partly or wholly precipitated with the copper, and must consequently be determined analytically in the deposit. A solution of ·970 gram copper, ·030 gram bismuth, gave 97·9 per cent. copper instead of 97 per cent.

"*Lead*, derived from the resolution of sulphate of lead (if present) by the wash-water, is partially precipitated with the copper. This applies only to large percentages of lead.

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* Mr. Sperry's experience shows that with proper precautions, these unfavorable results may be completely avoided.

"*Zinc and Nickel* do not interfere in quantities up to 30 per cent.

"*Arsenic* precipitates partly *with the copper*, and not *after it*, as has been supposed. It gives a bright deposit, but may be found in considerable quantity in the precipitate, before the solution is free from copper. After complete precipitation of the copper, therefore, the deposit should be titrated with cyanide of potassium."

Feeling that the important part played by the battery assay in modern copper metallurgy calls for a more detailed and perfect description than can be given by one whose assaying days are long past, I turned naturally to a chemist whose daily work lay in that direction, and was fortunate in securing the aid of Mr. Francis L. Sperry, analytical chemist, and for five years chemist to the Canadian Copper Company at Sudbury, Ontario.

The following description of the battery assay must therefore be credited to Mr. Sperry.

The notes on the electrolytic determination of copper are made up from the experience of five years' constant use of the method.

The scheme, as given, is intended to present the details in as practical and concise a manner as possible without going beyond the province of this work. Those who desire to study more closely the electrolysis of other metals, and also the treatment of copper in oxalate solutions which can advantageously be made use of, are referred to the admirable work of Dr. Classen on "Quantitative Chemical Analysis by Electrolysis," and also "Electro-Chemical Analysis," by Professor Edgar F. Smith.

THE DETERMINATION OF COPPER BY ELECTROLYSIS.

Of the various methods the chemist has in hand for the determination of copper, the electrolytic method presents some advantages over other recognized forms. It permits of reliable, clean, and rapid work, and enables the chemist to remove copper from a solution completely, in the presence of other metals, which may subsequently be determined in the same solution.

The requirements are clean platinum cathodes and anodes and a uniform current of electricity of known strength.

Take, for a weighed sample, one-half a gram copper matte, one or two grams copper ore, depending on the richness of the ore, and two or three grams for slag.

In preparing the samples they should be sifted through an eighty-mesh sieve. Weigh out on an accurate chemical balance.

After weighing the samples in duplicate, on watch glasses, transfer carefully to No. 2 beakers, slightly moisten with cold distilled water, add twenty-five c. c. strong nitric acid and ten to fifteen drops of strong sulphuric acid. The beakers should be covered with watch glasses and set on the sand bath, where they are heated until the nitrous acid fumes have all passed off and the sample is in solution. Wash the watch glasses down into the beaker, and evaporate the solution to dryness.

When choking white fumes appear, set the beakers one side to cool. The copper is now in the form of sulphate. Moisten the mass in the beakers with dilute nitric acid (1·20 sp. gr.), using about six or seven c. c.; add four drops of sulphuric acid, forty c. c. of water, and heat on sand bath until the mass is in solution. Filter off the insoluble matter (which should be examined to see if there may be copper left in the residue, undissolved), reserving the filtrate in a No. 1 beaker. The solution is now ready to be electrolyzed.

The electrical energy necessary to electrolyze a copper solution is furnished by various batteries of reliable manufacture. If there is an electric light plant at hand the wire, properly insulated, can be run through the laboratory, and by means of a resistance coil the current can be reduced in strength sufficiently to permit of quantitative electrolytic determinations. The Grenet, Gravity, or Grove cell batteries will be found well adapted for generating the necessary strength of current also. The Grenet cell loses its intensity after long use. The Gravity cell is very likely to act unsatisfactorily on account of local action setting in, causing polarization of the electrodes, and the electrical energy ceasing entirely. The Grove cell requires more care than either of the others spoken of, but the electromotive force is certain to act for as long a time as is necessary for the deposition of the metal, as the copper solutions are set on the battery at night and removed on the following morning, usually.

It is best to have a surplus of electrical energy for the electrolysis, although too strong a current must be guarded against.

Three Grove cells, freshly made up, will furnish current sufficient to electrolyze six to eight copper solutions, none of which contain more than ·5 gram copper in 1 gram of sample.

Four cells will electrolyze eight to ten solutions, and five cells ten to twelve solutions.

A convenient arrangement for supporting the cathodes and anodes for as many as twelve simultaneous determinations of copper is shown by the illustration (Fig. I.).

The rods are $\frac{3}{8}$ inch square by 39 inches long. Holes for the insertion of anode and cathode rods are $3\frac{1}{8}$ inches apart and $\frac{1}{16}$ inch in size, while through the side of the brass rods a milled screw sets against a flexible brass shoe, which binds the cathode and anode platinum rods securely in position. The brass rods, $\frac{1}{8}$ inch apart, are supported on glass pillars, and can be raised or lowered as required.

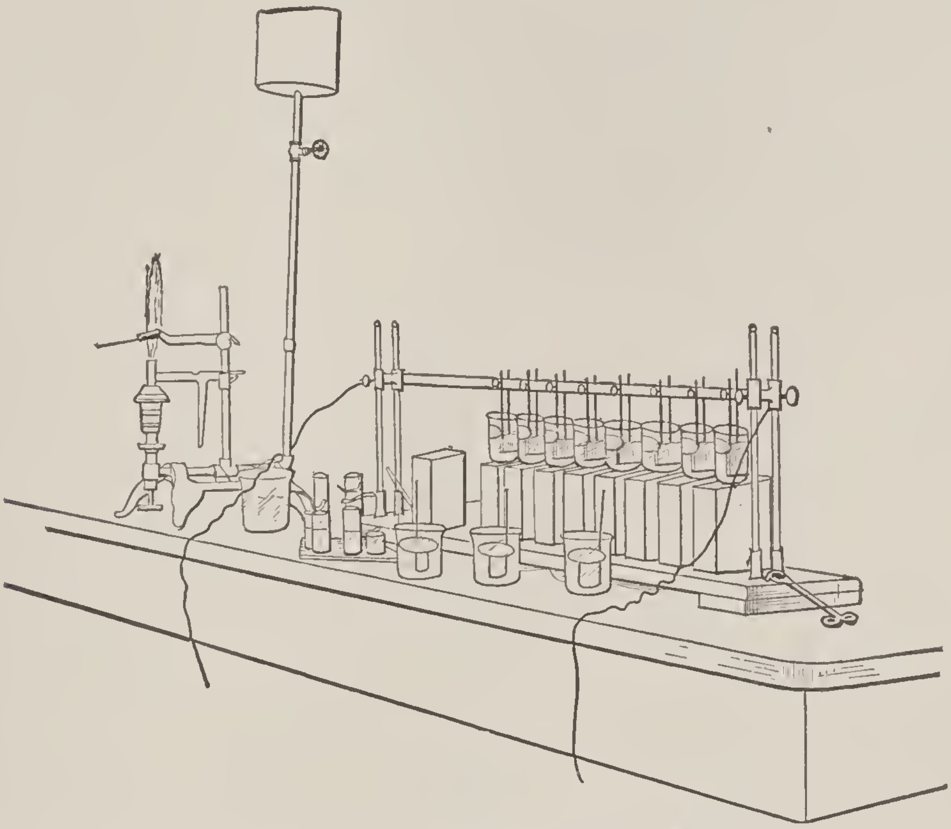

FIG. I.—RACK FOR BATTERY ASSAY.

The most convenient form of cathode is a plain platinum cylinder $2\frac{1}{2}$ inches long, 1 inch diameter, and the rod that supports it is $4\frac{1}{2}$ inches long. It weighs about 16 to 18 grams. (Fig. II.)

These cathodes may seem large, but for general class of work on high and low grade copper ores and mattes they will be found a very convenient size, as they offer a large surface for the deposition of copper, whereas, if they were smaller, frequently the copper would be deposited in spongy form, and there would be loss in weighing.

The anodes are platinum wire of size to fit $\frac{1}{16}$ inch hole, made in the form of a concentric circle, from the center of which the rod stands out 7 inches. (Fig. III.)

The diameter of the coil is 1 inch. By this arrangement of the anode there is a uniform evolution of gas throughout the solution, and the inside as well as the outside of the cathode is evenly electroplated with copper.

The cathode should not be completely immersed in the solution to be electrolyzed. When it is supposed that all the copper is deposited, immerse the cathode deeper in the solution and let the current run one-half hour longer. Any deposition on the clean surface will show at once that copper remains still in the solution. If the copper is all deposited, loosen the anode and carefully remove it and the beaker. Wash the cathode quickly into a clean No. 3 beaker with distilled water, immerse in pure alcohol, and gently ignite in flame until dry. The copper should be a pink rose color. Weigh as soon as cooled to the temperature of the room.

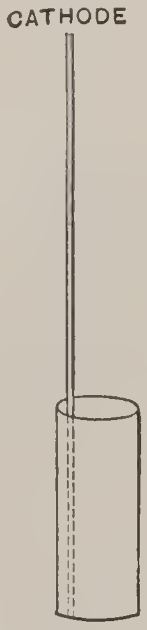

FIG. II.

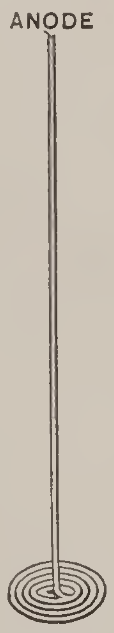

FIG. III.

FIG. IV.

The current should not be passed through the solution longer than is necessary to effect the complete deposition of the copper, as secondary reactions are liable to set in.

When there has been a separation of copper in a nitric acid solution alone, the solution should be siphoned off into a clean beaker without interrupting the current, and the cathode washed with pure water, otherwise the nitric acid will dissolve some of the deposited copper into the solution again. Too much nitric acid will keep the copper in solution. Too much sulphuric acid will cause the copper to deposit in spongy form.

Too strong a current will cause loss by too great evolution of oxy-hydrogen gas, the copper will deposit dark colored, and if zinc is present it will deposit on the copper.

By using deep beakers (Fig. IV.), there will be scarcely a perceptible loss of solution by the evolution of gas, as the sides of the beaker should be carefully washed down half an hour before removing the cathode to weigh.

The secondary reactions to be guarded against in passing the current longer than is necessary to deposit the copper, and also in not having the solution of proper strength of acid, are the conversion of the nitric acid into ammonia and the formation of ammonia sulphate, so that if the deposition of copper were done in the presence of iron and zinc, these metals would be deposited on the cathode as hydrated oxides.

With the conditions described above conformed to, copper is completely deposited and removed from solutions containing iron, alumina, manganese, zinc, nickel, cobalt, chromium, cadmium, lime, barium, strontium, and magnesium.

In the laboratory of the writer it has been customary to make electrolytic separations of copper and nickel daily for the past five years, and in every case with unvarying accuracy. The copper was removed completely in the presence of nickel, iron, and zinc, and these elements subsequently determined in the same solution.

Examples could be given *ad infinitum*, but a few will be given of the most characteristic.

	1 gm.	Sample taken.	
		1	2
SLAG................	Copper,	0·42%	0·42%
	Nickel,	0·41%	0·40%

	1 gm.	Sample taken.	
		1	2
COPPER ORE.........	Copper,	8·44%	8·43%
	Nickel,	3·33%	3·35%

	1 gm.	Sample taken.	
		1	2
NICKEL ORE.........	Copper,	1·23%	1·24%
	Nickel,	8·62%	8·63%

	.5 gm.	Sample taken.	
		1	2
COPPER MATTE......	Copper,	33·45%	33·46%
	Nickel,	15·75%	15·73%

	.5 gm.	Sample taken.	
		1	2
NICKEL MATTE......	Copper,	16·76%	16·78%
	Nickel,	21·23%	21·25%

In each case the nickel was determined electrolytically in the same solution from which the copper was removed.

By carefully noting what are the best conditions, as there is a certain limit within which variation in the treatment of miscellaneous samples is warranted, most any novice will find electrolysis a simple and accurate method for the estimation of copper.

CHAPTER IV.

THE ROASTING OF COPPER ORES IN LUMP FORM.*

ROASTING or calcination, used indiscriminately in the language of the American copper smelter, signifies the exposure of ores of metals containing sulphur, arsenic, and other metalloids to a comparatively moderate temperature, with the purpose of effecting certain chemical, and rarely mechanical, changes required for their subsequent treatment. This definition is restricted to the dry metallurgy of copper, and does not take into consideration chloridizing roasting, roasting with sulphate of soda, and other well-known variations, which belong either to the metallurgy of the precious metals or to the wet treatment of copper ores.

The care and attention which should be devoted to this preparatory process cannot be too strongly insisted on, nor can any one carry out either this apparently simple roasting or the following fusion to the best advantage, who is not thoroughly familiar with the striking chemical changes that in every calcination follow closely upon each other, and by which the sulphides and arsenides of the metals are transformed at will into a succession of subsulphides, sulphates, subsulphates, and oxides. These, reacting upon each other according to fixed and well-known laws, enable the metallurgist at his pleasure to produce every grade of metal from black copper to a low-grade matte that shall contain nearly all the metallic contents of the ore in combination with sulphur. To avoid constant repetition, it may be understood that in speaking of calcination, when sulphur is mentioned, its more or less constant satellites, arsenic and antimony, are also included, their behavior being somewhat similar under ordinary circumstances. These very different products, as well as the amount of ferrous oxide, the most important basic element of every copper slag, result solely from the degree to which the calcination is carried. In fact, it may be taken as literally true, that the composition of both the valuable and waste products of the fusion of any sulphide ore of copper is determined irrevocably and entirely in the roasting-furnace or stall. A more thorough study of the reactions just referred to will be found in its proper place. Enough has here been said not only to explain the author's object in devoting so much attention to this process, but also to induce such metallurgists as are not already thoroughly familiar with the theory of calcination to endeavor to become so if they desire to ever excel in the economical treatment of sulphide ores.

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* In English metallurgical literature, the term roasting is applied exclusively to that process in which copper matte in large fragments is exposed on the hearth of a reverberatory furnace to an oxidizing atmosphere, and a moderate, but gradually increasing temperature. See "Matte Concentration."

The varieties of calcination, as applied to the dry treatment of copper ores, are at most two:

1. The oxidizing roasting, which is necessarily combined with volatilization.

2. The reducing roasting, limited in its application almost exclusively to substances containing much antimony or arsenic.

Plattner's admirable work on *Röstprocesse* contains the whole theoretical part of calcination; but a foreign language is a barrier to many ardent students of metallurgy, and his descriptions and plans of furnaces and apparatus apply to those in use during the past generation. A modern treatise on roasting, regarding the subject principally from a practical standpoint, and adapted to present American conditions, seems desirable. Such a treatise, however, could not attain the highest degree of usefulness without a consideration of the theory of calcination sufficient to enable and encourage all who make use of the more practical part to follow with ease the chemical reactions on which the process is based. These reactions can be more easily appreciated and remembered after first becoming familiar with the apparatus and means by which the exposure of the substances under treatment to the influence of heat and air is effected. The theoretical discussion will be postponed until this familiarity is attained.

This roasting apparatus must vary according to the mechanical condition of the ore under treatment—that is, whether fine or coarse. (See article by the author, "The Roasting of Copper Ores and Furnace Products." *United States Geological Survey, Mineral Resources of the United States*, Albert Williams, Jr., 1883.) It is assumed that the process of calcination, as executed in the dry metallurgy of copper, has to deal with the oxidizing of only two classes of material, ores and mattes. The appliances for the roasting of these substances in lump form may be divided into three classes:

1. Heap roasting. Suited to both ore and matte.
2. Stall roasting:
    a. Open stalls. Suited only to ore.
    b. Covered stalls. Suited to both ore and matte.
3. Kilns. Suited only to ore.

The mechanical preparation of the material for each of these three forms of roasting is virtually identical, and has an important influence on the result of the process. The size to which the substance under treatment should be broken cannot be arbitrarily stated, as different ores vary so greatly in their composition and behavior. Ores containing a high percentage of sulphur —twenty-five and over—will give excellent results if so broken that the largest fragments shall be capable of passing through a ring three inches in diameter; while more rocky ore, which is likely to be of a harder and denser texture, should be reduced to pass a two-inch opening. Careful experiments can alone determine the most profitable size for any given material, and should be continued on a large scale until the metallurgist in charge has fully satisfied himself on this point. This may be determined with the least trouble and expense by noticing the weight and quality of the matte obtained by smelting the roasted ore from various heaps formed of fragments differing in their maximum size.

All other conditions being identical, the heap that yields the smallest quantity of the richest matte has, of course, undergone the most perfect oxidation, and should be selected as a standard for future operations. Variations that may occur in the chemical or mechanical condition of the ore should be carefully watched as a guide in fixing upon the best roasting size. Local conditions must determine whether a jaw-crusher or hand labor should be used for this purpose. The production of fines is a decided evil in the preparation of ore for heap roasting, and the manual method possesses a certain advantage in this respect, though this consideration may be outweighed by other economic conditions. A trial of the comparative amount of fines produced by machine and hand-breaking was carried out on three different varieties of sulphureted copper ores of average hardness, and aggregating 2,220 tons.* The broken ore was thoroughly screened; all passing through a sieve of three meshes to the inch (6 mm. openings) was designated as fines.† One-half (1,110 tons) of this material was passed through a seven by ten jaw rock-breaker, with corrugated crushing-plates (which produce a decidedly less proportion of fines than the smooth plates). The breaker made 240 revolutions per minute, and had a discharge opening of two and one-half inches. The other moiety was broken by experienced workmen, with proper spalling-hammers, into fragments of a similar maximum size. The result was as follows, only the fine product being weighed, the coarse being estimated by subtracting the former from the total amount:

Broken by Jaw-Crusher.

	Tons.	Per cent.
Fine product—below 6 mm. in diameter.....	192·25	17·32
Coarse product—between 6 mm. and 64 mm.	917·75	82·68
Total..............................	1,110·00	100·00

Broken by Hand Hammers.

	Tons.	Per cent.
Fine product—below 6 mm. in diameter.....	103·34	9·31
Coarse product—between 6 mm. and 64 mm.	1,006·66	90·69
Total..............................	1,110·00	100·00

These results are quite in accordance with impressions derived from general observation, and, as will be noticed, prove that, with certain classes of ore, mechanical crushing produces nearly double as much fines as hand-spalling. As 10 per cent. of fines is an ample allowance to form a covering for any kind of roast-heap—and better results are obtained when the same partially oxidized material is used over and over again as a surface protection—it may frequently occur that, in spite of its greater cost, hand-spalling may prove more profitable than machine-breaking. This is a matter for individual decision, and can be determined only after a mature consideration of the difference in expense of the two operations, the means at hand for the calcination, and subsequent advantageous smelting, of the increased quantity of fines, and whatever other factors may bear on the case in hand. The following steps should be carried out, whichever method is decided upon. The ore, after breaking, should be separated into three classes, the largest including all the product between the maximum size and one inch (25 mm.); the medium size, or ragging, consisting of the class between 25 mm. and the fine size (three meshes to the inch, which would give openings of about 6 mm. net); and the fines, as already explained. Roughly speaking, the percentage of each class, including the fine ore that is invariably produced during the operation of mining, may be represented by the following figures:

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* Unless otherwise indicated, all tons equal 2,000 pounds.

† It should also be explained that, owing to the large and very variable amount of fine material contained in the ore before crushing, as it came from the mine, it was passed over the screen just referred to before being either fed to the crusher or spalled by hand. Without this precaution, the results of the trial would have been valueless, as the variation in the amount of fines in the original ore was far greater than the discrepancy in the amount produced by the two different methods of crushing.

Coarse	55	per cent.
Ragging	25	"
Fines	20	"
Total	100	"

This classification is effected with great ease and economy in case machine-breaking is decided upon, by the use of a cylindrical or conical screen of $\frac{3}{16}$-inch boiler iron, about 8 feet in length and 36 inches or more in diameter. This is placed below the breaker so that it receives all the ore. It is made to revolve from 18 to 22 times per minute, and has a maximum fall of an inch to the foot. This can easily separate 10 tons of ore per hour, and by proper arrangement of tracks or bins, discharge each class into its own bin. One fault in this very simple classifying apparatus is, that the coarse lumps of ore must necessarily traverse all the finer sizes of screen, thus greatly augmenting the wear and tear. This objection, though frequently valid under other circumstances, has but little weight when it is remembered that even the smallest holes (6mm.) are punched in iron of such thickness ($\frac{3}{16}$ inch) that it will withstand even the roughest usage for many months. To produce the three sizes just alluded to, the screen requires two sections, with holes respectively 6 mm. and 25 mm., of which the finer size should occupy the upper 5 feet, and the coarser the lower 3 feet of the screen. In remote districts, where freight is one of the principal items of expense, heavy iron wire cloth may be substituted for the punched boiler iron, and if properly constructed and of sufficiently heavy stock, will be found satisfactory, lasting about one-half as long as the more solid material. The difference in size between a circular hole 25 mm. in diameter and a square with sides of that length, should not be overlooked in changing from one variety of screen to the other. The mouth of the crusher should be level with the feeding-floor, and the latter should be covered with quarter-inch boiler iron, firmly attached to the planks by countersunk screws, by which arrangement the shoveling is greatly facilitated. With such a plant, two good laborers will feed the breaker at the rate of ten tons an hour for a ten-hour shift, provided none of the rock is in such masses as to require sledging, and that the ore is dumped close to the mouth of the breaker. A seven by ten jaw-breaker of the best and heaviest make is capable of crushing the amount just mentioned to a maximum size of 2½ inches, provided the rock is brittle, heavy, and not inclined to clog the machine. In most cases where this duty is required, and especially if the ore is damp and in large fragments, it is much more advantageous to substitute a fifteen by nine breaker, which, when geared up to 230 revolutions and with sufficient power, has a capacity of at least twenty tons per hour on favorable ores, and when properly fed.

The expense per ton of breaking, sizing, and delivering into cars with such a plant operating upon ores of medium tenacity, is as follows, the figures being deduced from average results of handling fully one hundred thousand tons under the most varying conditions. It is assumed that the breaker is run by an independent engine of sufficient power,* while the wages of an engineer and firemen are partially saved by taking the steam from the boilers that are supposed to supply the main works:

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* Speaking from a very extensive experience, the author finds that not one breaker in ten is run up to anything approaching its capacity, and that consequently it has become customary to provide an engine and boiler far too small to drive the breaker up to speed when doing full work. The statements made above refer to breakers run to their extreme capacity, and under these conditions a 7 by 10 crusher requires an engine of not less than 6 by 10 cylinder, while a 15 by 9 crusher requires an 8 by 12 cylinder. As each movement of the jaw crushes a given amount of ore, it follows that the capacity of a breaker depends upon the speed at which it is run.

COST OF BREAKING ORE BY MACHINERY WITH A PLANT OF 100 TONS CAPACITY IN TEN HOURS.

		Per 100 tons.	Per ton.
Power—per day of 10 hours:			
1,200 pounds of coal at $4.50 per ton	$2.70		
Oil and lubricants	.40		
Engineer, ¼ wages at $3	.75		
Fireman, ¼ wages at $2	.50	$4.35	$0.0435
Labor:			
Two feeders at $1.75		3.50	0.0350
Repairs:			
Toggles and jaw plates, etc	$0.43		
Wear of tools, Babbitt for renewing bearings, etc	0.37		
Daily slight repairs on machinery	0.35		
Miscellaneous items, sampling, etc	0.33	1.48	0.0148
Sinking Fund:			
To replace machinery, at 10 per cent. on original cost		0.78	0.0078
Total		$10.11	$0.1011

If it should seem at first glance that 10 cents per ton is an unreasonably low figure, it will be noticed that the cost of transportation both to and from the breaker is not included in this estimate; the former is usually charged to mining expenses, and the latter to heap-roasting. Ore that is to undergo roasting in kilns for the purpose of acid manufacture must be broken considerably smaller than that just described, and this, of course, lessens the capacity of the apparatus and proportionately increases the expense. An increase of 50 per cent. in the above estimate will be sufficient to cover it. The figures given above have been frequently attained by the author, but only under certain favorable conditions, among which are: Abundance of power to run the breaker to its full speed, regardless of forced feeding. A constant system of supervision by which the plant is kept up to its full capacity of ten tons per hour, and which demands exceptionally good men as feeders. A frequent inspection of the machinery, and renewal of all jaw plates, toggles, and other wearing parts, before the efficiency of the machine has begun to be impaired; all of which repairs should be foreseen and executed during the night shift or on idle days. A perfect system of checking the weight of all ore received and crushed, without which precaution a mysterious and surprisingly large deficit will be found to exist on taking stock.* It is hardly necessary to mention that all bearings that cannot be reached while the machinery is in motion must be provided with ample self-oilers, and since clouds of dust are generated in this work, that unusual care must be taken in covering and protecting all boxes and parts subject to injury from this cause. Unless the ore is sufficiently damp—either naturally or by artificial sprinkling—to prevent this excessive production of dust, the feeders should be required to wear some efficient form of respirator; otherwise, they are likely to receive serious and permanent injury, the fine particles of sulphides being peculiarly irritating to the lungs and entire bronchial mucous membrane.

The breaking of ore by hand hammers, technically denominated "spalling," is worthy of more careful consideration than is generally bestowed upon it. The style of hammer is seldom suited to the purpose, though both the amount of labor accomplished and the personal comfort of the workmen depend more upon the weight and shape of this implement and its handle than on any other single factor save the quality of the ore itself. There should be several cast-steel sledges, differing in weight from 6 to 14 pounds, and intended for general use in breaking up the larger fragments of rock to a size suitable for the light spalling-hammers. Each laborer should be provided with a hammer 6 inches in length, forged from a 1½ inch octagonal bar of the best steel, and weighing about 2¾ pounds. This should be somewhat flattened and expanded at the middle third, to give ample room for a handle of sufficient size to prevent frequent breakage. The handles usually sold for this purpose are a constant source of annoyance and expense, being totally unsuited to this peculiar duty. It is better to have the handles made at the works, if it is possible to procure the proper variety of oak, ash, hickory, or, far better than all, a small tree known in New England as iron-wood or hornbeam, which, when peeled and used in its green state, excels any other wood in toughness and elasticity. The handles should be perfectly straight, without crook or twist, so that, when firmly fastened in the eye of the hammer by an iron wedge, the hammer hangs exactly true. Their value and durability depend much upon the skill with which the handles are shaved down to an area less than half their maximum size, beginning at a point some six inches above the hammer-head and extending for about ten inches toward the free extremity. If properly made and of good material, they may be made so small as to appear liable to break at the first blow; but in reality they are so elastic that they act as a spring, and obviate all disagreeable effects of shock; wear longer and do more work than the ordinary handle. Such a handle has lasted five months of constant use, in the hands of a careful workman, whereas one of the ordinary make has an average life of scarcely four days, or perhaps thirty tons of ore. Where the ore is of pretty uniform character, it is advantageous to adopt the contract system for this kind of work. A skillful laborer, under ordinary conditions, will break seven tons of rock per ten-hour shift to a size of 2½ inches,* taking coarse and fine as it comes, and in some cases he is also able to assist in screening and loading the same into cars. This latter operation should be executed with a strong potato-fork having such spaces between the tines as to retain the coarsest size, while the finer classes are left upon the ground. These forks are made for this purpose by a firm in St. Louis, and are much superior to the ordinary forks. When a sufficient quantity of the finer classes has accumulated and the pile or stall is ready to receive its outer layer of ragging, the mixed material should be thrown upon a screen inclined to an angle of about 48 degrees and having three meshes to the inch. This screen is elevated upon legs to such a height that the coarser class that fails to pass its openings will be caught in a car or barrow, while the fines fall either into a second movable receptacle or upon the floor, being in the latter case prevented from again mixing with the unscreened ore by a tight boarding on the front and sides of the screen frame. The amount of space required for convenient spalling is about forty square feet per man, which will allow for ore-dumps, tracks, sample boxes, etc. A good light is essential, especially if any sorting is to be done, and it is in this case and where fuel is expensive that hand spalling frequently presents especial advantages. When the ores are siliceous, a mere rejection of such pieces of barren quartz or wall rock as have accidentally got among the ore, or first become visible on breaking up the larger masses, may have a most beneficent influence on the subsequent fusion. Where the expense of treatment is high and work is conducted on a large scale, the profit resulting from raising the average contents of the ore even a single per cent. is hardly credible, even aside from the increased fusibility due to the diminished proportion of silica. To illustrate: At certain works that the author was called to superintend, it had been the custom to spall all the first-class ore just as it came from the mine without any sorting out of barren wall rock, considerable quantities of which were mixed with the ore. Fuel and labor were very high, and the ore mixture already too siliceous. A rough method of sorting was instituted, and some twelve per cent. of the entire weight of the first-class ore was thrown out with the loss of scarcely any metal. The month's average assay of ore due solely to this sorting was increased 2½ per cent., and the furnaces gave an extra yield of 1,500 pounds of copper from 30 tons daily, or 45,000 pounds for the month, which, calculated on the spot at 10 cents per pound, was a gain of $4,500. The net gain was probably even more than this; for the expense of sorting was hardly appreciable, while the increased fusibility of the charges, and the fact that 3,000 pounds of barren material could be replaced by an equal amount of good ore, added largely to the profits.

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* This is a difficulty that the metallurgist will encounter at every stage of his work. In spite of the most accurate scales, and of careful and frequent determinations of weights, the quarterly balance-sheet will invariably show that the actual amount of ore treated is less than the amount shown by the weigh-master's book; while the weights of all supplies consumed, especially fuel, have been reported too low. To one unprepared for this result, the consequences may be serious.

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* Unless otherwise specified, the term "day" or "shift" may be understood to signify the ordinary working day of ten hours, from seven A.M. to six P.M., with one hour for dinner.

All windows in the spalling-shed must be protected by strong iron wire netting, three meshes to the inch; nor should the eyes of the workmen receive less care than the panes of glass. Accidents from flying fragments of sharp rock are common, and frequently result in a partial or total loss of vision, which entails serious expense on the company, and is an infliction almost worse than death upon the victim. All this can be easily avoided by the use of wire goggles, strongly and properly made, so that while completely protecting the visual organs, they cause but little annoyance to the wearer. These should be furnished by the employer, and should be constantly worn, on pain of dismissal. The workman, with proverbial recklessness, will sometimes claim that he has a right to risk his own eyes if he chooses; but the employer may demand the privilege of protecting himself against those claims which, with more or less reason, are sure to be made in case of injuries received while in his employ. Artificial warming of the building is neither necessary nor desirable. Lithe, active men or boys, of nervous temperament and quick, accurate movements, should be selected for this work, which calls rather for rapidity and knack than for any great muscular effort. The amount of rock broken being about proportionate to the number of effective blows delivered, it follows that, other things being equal, a man who delivers twenty blows per minute will accomplish nearly double the work of one whose deliberate temperament would naturally limit his motions to half that number during the same time. It is just in this matter of selecting workmen adapted to each variety of labor that long experience in the management of men, and a thorough knowledge of human nature, enable one man to obtain results and effect improvements that seem well-nigh impossible to him who is unaccustomed to such a perfect adaptation of means to ends.

The cost of spalling an ore of the same character as that on which the foregoing estimates for machine-breaking are based has been calculated from the average results of a very large quantity of ore, assuming 100 tons to be spalled, screened, and loaded in ten hours.

COST OF SPALLING ORE BY HAND WITH AN OUTPUT OF 100 TONS PER TEN HOURS.

Labor :		Per 100 tons.	Per ton.
14 men breaking ore, including screening and loading, at $1.50	$21.00		
4 men sledging and loading at $1.50	6.00		
1 foreman	2.50	$29.50	$0.295
Repairs :			
Including new steel and handles.			
5 handles at 30 c	1.50		
7 pounds of steel at 15 c	1.05		
Blacksmith's and other work on above, $\frac{1}{3}$ day	1.00		
Screens, forks, and shovels	1.67		
General repairs	0.55	5.77	0.0577
Sinking Fund:			
To replace screens and permanent fixtures		0.15	0.0015
Total		$35.42	$0.3542

This is about 25 cents per ton greater than by machine-breaking. The same addition—50 per cent.—will here also cover the increased cost of breaking the ore smaller for kiln-roasting.

I. HEAP-ROASTING.

The roasting of sulphureted ores or copper in mounds or heaps dates back beyond the age of history, and in its most primitive form is still practiced among barbarous nations who have evidently never held communication with each other. It is not difficult to imagine its origin in the midst of some rude people, whose possession of superficial deposits of oxides and carbonates of copper had taught them the value of that metal as obtained by a simple process of fusion, while the sulphide ores that were doubtless encountered at a slightly greater depth were thrown aside in heaps as worthless until the spontaneous combustion of some of these waste-piles, brought about by the decomposition of the sulphides, and the interesting discovery that ores, hitherto considered valueless, would, after a simple burning, also yield the coveted metal, led some metallurgist of that day to the idea of calling in the aid of artificial combustion to hasten matters. Nor has this rude and simple process undergone that general improvement that one might have expected when considering the tremendous advances made in other appliances for accomplishing the same purpose. A somewhat careful inspection of nearly all the localities in the United States where heap-roasting is practiced reveals the fact that the results obtained are far from satisfactory in the greater number of instances. The amount of fuel employed and the height and size of the heap are not correctly proportioned to the sulphur contents of the particular ore under treatment. Fragments of rock far exceeding in size the extreme proper limit, as determined by experience, are mixed with material so fine as to be fitted only for the covering-layer, and these are dumped upon the ill-arranged bed of fuel without regard to the final shape of the structure or the establishment and maintenance of the requisite draught. Also, a sufficient quantity of proper material for the all-important covering layer is not applied. The result of these and some other deficiencies is that a small proportion only of the ore is exposed to a proper degree of heat, and the remainder of the heap is pretty equally made up of half-molten masses of clinkers from the interior, and comparatively raw and unburned material from the outer layer. With the exception of what little sulphur may have been driven off by volatilization, the ore after such a calcination is scarcely better fitted for the fusion that is to follow than if it had not been roasted. The evil results of an imperfect preliminary calcination can only be fully appreciated after the ore has passed to the next stage of treatment; in fact, they are so far-reaching, that it is impossible to express the full measure of the damage in exact figures. A discussion of the effect of imperfect calcination and of its remedies will be found under the head of "Smelting Sulphide Ores in Blast-Furnaces." The vital importance of the process, and the almost universal want of care and supervision in the carrying out of its details, will justify this urgent remonstrance against its improper execution. Moreover, the cost of roasting properly is no greater than that of doing it imperfectly.

The responsibility of selecting *heap-roasting* in contradistinction to the other methods enumerated for the desulphurization of an ore must rest upon the metallurgist in charge of the works, and is a question deserving the most careful consideration; nor are the reasons for or against its adoption in most cases so clear and self-evident that plain and unvarying rules can be laid down for his guidance. In this, as in many other instances, there are usually strong metallurgical, commercial, and sanitary arguments that should be carefully weighed. The contiguity of cultivated land, or even of valuable forests, would forbid the employment of heap-roasting unless the arguments for its adoption were sufficiently powerful to outweigh the annoyance of constant remonstrances on the part of the land-owners, accompanied by claims for heavy damages from the effect of the sulphurous gases. For legal reasons, as well as for various other prudential and sanitary motives, it is important to learn how this damage is effected, and to what distance its ravages may extend.

1. The damage is caused solely by sulphurous and sulphuric acids, neither arsenical nor antimonial fumes nor the thick clouds of smoke evolved from bituminous coal having any appreciable influence.

2. The most injurious effects are visible on young, growing plants; and the more tender and succulent their nature, the more rapid and fatal are these.

3. A moist condition of the atmosphere greatly heightens the

injurious effects of the gases, and as our most frequent rains occur in the spring, at the very period during which the crops and forests are in young, green leaf, more damage may be effected in a few days at this season than during the entire remainder of the year. The author has seen a passing cloud, while floating over a dozen active roast piles, absorb the sulphurous smoke as rapidly as it arose, and, after being wafted to a distance of some eight miles by a gentle breeze, fall in the shape of an acrid and blighting rain upon a field of young Indian corn, withering and curling up every green leaf in the whole tract of many acres in less than an hour.

4. As might be expected, the vegetation nearest the spot where the fumes are generated suffers the most, and the direction of the prevailing winds, in a fertile district, can be plainly determined by the sterile appearance of the tract over which they blow.

The most elaborate means for obviating this evil have been tried at the great metallurgical establishments of Europe, and vast sums have been expended in this direction. The plans pursued in England tend more toward the mechanical deposition of the offending substances in long flues and passages (the first experimenters evidently having failed to realize that the sulphurous vapors alone caused the damage), while in Germany, the more scientifically correct method of effecting condensation and absorption of the gases by means of various liquids and chemicals was pursued, but with scarcely better results. In the former case, it was soon discovered that, while the oxides of zinc, lead, arsenic, antimony, and various other substances carried over mechanically or as gases by the draught, were condensed and deposited so completely in the canals that the air issuing from the top of the tall chimney was practically free from them, the percentage of sulphurous and sulphuric acids, which alone are responsible for damage to vegetation, was not sensibly diminished. Similar efforts in Germany for the absorption of the sulphur gases were carried out with such imperfect and ill-adapted apparatus, and on so inadequate a scale, that the absolute impossibility of a successful issue must be apparent to any one reading the pamphlet issued by the Freiberg officials intrusted by government with the execution of the experiments. But however insufficient the apparatus, the results arrived at decisively indicated the impossibility of disposing of the offending fumes by any plan of condensation or chemical absorption, except on a small scale and with unusually dilute gases.

The problem has long been solved in Europe in the only rational and economical manner, by utilizing the hitherto destructive fumes for the manufacture of sulphuric acid. This requires, of course, the abolition of heap-roasting, and the confinement of all processes of calcination to such closed kilns and furnaces as may be placed in direct communication with the leaden acid chambers. The very secondary position held by agriculture in those sections of our country that furnish the material for the principal smelting-works has, up to the present time, obviated any necessity of dealing with this question, though some of the largest copper smelting-works in the East have already adopted the European solution of the problem as a matter of profit rather than necessity.

In the case of smelting establishments of such capacity that not more than twenty-five tons daily of sulphur are oxidized and poured into the atmosphere, it is probable that all vegetation outside of a circle of four miles in diameter may, under ordinary circumstances, be considered safe from the effects of the fumes.

No harm to man or beast has ever been authentically reported as resulting from the use as food of an article of vegetable origin that has been exposed to the corrosive influence of such gases. This is a very important point, and careful investigation and experiments have completely disproved the opposing arguments so often made against smelting-works in Germany by certain stock-raisers.

During the preparation of the new edition of this book, Butte City, Montana, is anxiously seeking a remedy for the vast clouds of sulphurous vapors that daily arise from the treatment of its rich, sulphureted copper ores. By a rough estimate, there are probably not less than 200 tons daily of sulphur thus oxidized and escaping into the atmosphere, forming a nuisance that must be experienced to be appreciated.

As I have already intimated, several carefully-formed European commissions have investigated the question of handling these fumes with more care and skill than is likely to be brought to bear upon the matter by any private enterprise, however liberally endowed.

The result of their investigations indicated that the only rational method of combating this nuisance was to utilize the noxious gases that had hitherto been allowed to escape into the atmosphere, and for this purpose, immense lead chambers were built, and nearly all the calcining apparatus arranged to discharge their fumes into the latter, in which they were converted into sulphuric acid by the usual means; the result being not only a total cessation of the nuisance, but a quite satisfactory profit therefrom.

At Butte City, owing to its isolation and consequent high freight rates to a market, the direct manufacture of sulphuric acid could not be undertaken with any hope of profit.

From a personal knowledge of the conditions that prevail in Montana, and after careful consideration of the same, I can see only one feasible and economical plan for the abatement of the fumes. I should suggest:

1. An entire abolition of heap-roasting, substituting therefor stall-roasting, or kiln-roasting, according to the character of the ore.

2. Such an arrangement of furnaces and flues that the sulphur fumes may be obtained in as concentrated a form as possible, to be used as described in paragraph No. 3.

3. The utilization of these sulphurous acid gases for precipitating the copper as a dichloride, and eventually producing it in a metallic form from a solution obtained by calcining and dissolving the copper-silver mattes, which form the main product of all the smelting-plants, and are now mostly shipped to Europe for separation and refining. By the new Hunt & Douglas method, the fumes are thus converted into sulphuric acid without the necessity of any costly lead chambers, or the ordinary cumbersome and expensive paraphernalia of an acid-plant, and thus used directly in treating the mattes, and obtaining a copper of the highest quality, which by one simple refining operation is put into ingot equal to Lake copper for most purposes. The silver and gold are obtained from the residues by the ordinary methods.

4. The sulphur fumes arising from those operations which fail to produce a gas sufficiently concentrated for the purpose just indicated, should be discharged into the upper regions of the atmosphere by means of very high chimneys, as is done at all chemical works in the neighborhood of cities, and with per-

fect success; the fumes being so diluted with air before they reach the ground as to be absolutely harmless to vegetable life, as well as imperceptible to the human senses.

It has been satisfactorily demonstrated, by the evidence of long, practical trial, as well as by elaborate experiments, that so small a proportion of arsenic as exists in ordinary sulphureted fumes is perfectly harmless both to man and beast.

Within a few months, a New York firm has brought out a condenser, which claims to completely abate the fume nuisance. If it were not endorsed by first-class metallurgists, I could scarcely believe that so small an apparatus could produce so great a result. It is worthy of investigation.

In laying out the ground for roast-piles, the first point to consider is the prevailing direction of the wind, great care being taken that the fumes shall neither be blown toward the works themselves, nor toward the offices and dwelling-houses in their immediate neighborhood. Smelting-works are frequently situated in a valley, in which the prevailing winds naturally follow its longitudinal axis. In this case, a tract of ground on one side or other of the central depression, instead of in its immediate course, should be selected. By careful observation, and taking into consideration that the prevailing winds may differ at different seasons of the year, the roast heaps can generally be so placed as to give no substantial ground for claims of damage to agriculture. Care should also be taken that the selected tract is free from any possible chance of inundation; that it is either perfectly dry, or susceptible of thorough drainage; that it is not crossed by gullies or depressions that may serve as water-courses for the drainage of the surrounding hills in case of a heavy shower; that it is protected as far as possible from violent winds; that snow does not drift on it badly in winter, and that it is at least as high as the spot to which the ore is to be transported for the ensuing operation, or, if this is not feasible, at least as high as the elevator which is to raise it to the required level. If possible, it should occupy an intermediate position, as regards grade, between the shed in which the ore is prepared for roasting and the point at which the calcined product is to be delivered. A fall of ten feet for the first step and four and one-half or more for the second—total fourteen and one-half feet—will render possible the establishment of a system of handling and transportation that can hardly be excelled.

A detailed description of such a model plant will suffice as a pattern that may be varied to suit local conditions, always remembering that, under ordinary American circumstances, the economy of labor is one of the first conditions to be observed, and that the saving of 25 cents in handling a ton of crude ore is equal to a dollar or more on the ton of matte, and at least two dollars when estimated on the ton of copper.

Assuming that the metallurgist is called upon to prepare a yard for heap-roasting of ample size to contain a sufficient number of piles to furnish from 80 to 100 tons daily of calcined material, without encroaching upon the partially burned ore, and that the contour of the ground permits the requisite fall in each direction—as already explained—the following plan may be advantageously adopted:

Experience having demonstrated that an ordinary pile 40 feet long, 24 feet wide, and 6 feet high will contain about 240 tons, and burn for 70 days, to which should be added 10 days for removing and rebuilding, it follows that each pile will supply $\frac{240}{80} = 3$ tons of roasted ore daily; so that 35 heaps will be needed to furnish the full amount of 100 tons daily. Allowing 36 feet for the width of each structure, and 60 feet for the length, in order to give ample room for various purposes that will be explained hereafter, an area of 75,600 square feet will be required.

The frost being out of the ground and the surface dry, a rectangular area of the extent just computed should be prepared by means of plow and scraper, being leveled to a perfect plane, and having a slight slope toward one longitudinal edge, or from a central ridge toward either side. The black surface soil should be removed, together with all sods, stumps, and remains of vegetation, and the space that it occupied replaced with broken stones, slag, or coarse tailings from the concentrator; or, best and cheapest of all, granulated slag from the blast-furnace. This can be easily obtained in any desired amount by allowing the molten scoriæ from the slag-spout to drop into a wooden trough, lined with sheet iron, placed with a grade of one inch to the foot, and provided with a stream of water running through it, equal to at least sixty gallons a minute. If sufficient fall is available, the granulated slag—graduated to any desired size by the height through which it falls, velocity and amount of water, and various other trifling factors easily ascertained by trial—is discharged directly from the launder into dump-carts, the water being drawn off by substituting a sieve of ten meshes to the linear inch for the lower eighteen inches of the wooden trough bottom. By this simple means, the best kind of filling can be prepared and delivered at the roasting-yard for nothing, the expense of transportation hardly equaling the wages of the ordinary slag-men, who may be employed in attending to the loading of the carts and the leveling of the material when dumped. The entire area of the rectangle being raised at least two inches above the surrounding ground, a proper surface is formed by spreading upon the foundation already described a sufficient quantity of clayey loam. This should be rolled several times with a heavy roller drawn by horses, the surface being slightly dampened from time to time, until the entire area is as level and nearly as hard as a macadamized road.

Unless the climate is an unusually dry one, and the district free from snow, it will be better to use gravel instead of the loam, putting down a layer some four inches thick over the entire surface of the roast-yard. This will prevent mud, and the great loss arising from the treading of the fine ore into the same.

If the roast-yard is to be a permanency, and one is desirous of obtaining the best results with the least loss, a final covering of ore-fines should be added, the gravel being covered three or four inches deep with low-grade fines. Nor should this covering be confined merely to the portion of the ground that is to be occupied by the ore-heaps, but should be applied to the entire surface, including spaces between the heaps, passageways at ends of heaps, etc., etc. By so doing, there will always be a caked coating of ore-fines to shovel on, and the danger of getting dirt and gravel mixed with the roasted ore will be avoided completely.

As the layer of fines beneath the heaps becomes gradually roasted through, it should be removed with the coarse ore and sent to the furnaces, its place being supplied by fresh fines of the richest description, for nowhere can fine ore be roasted so free from any possibility of loss as when safely buried beneath the heap.

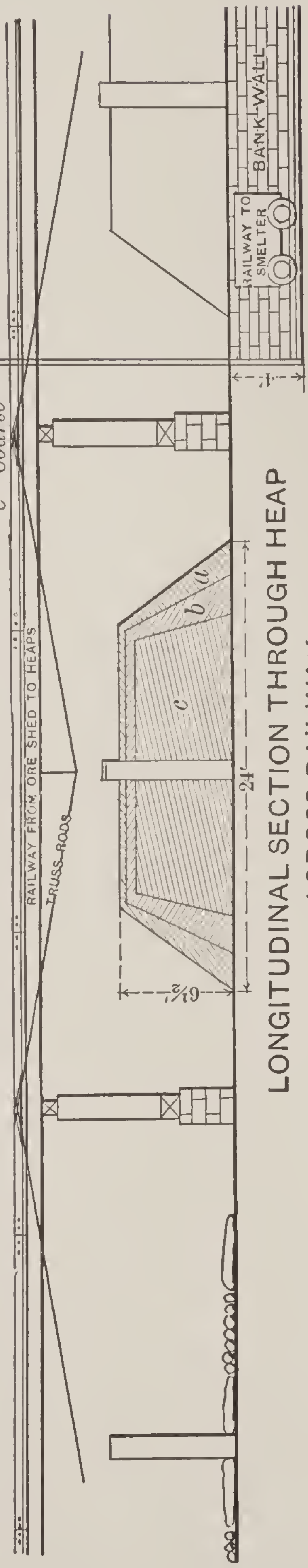
Nothing is more important about a roast-yard than a proper drainage system. If possible, the entire ground should slope slightly toward the lateral lower track on which the roasted ore is removed to the furnaces; and where such a gentle slope can be obtained, the drainage problem is rendered very simple and perfect; for a deep ditch run all along the upper edge of the mound, parallel with the track just referred to, will cut off all the surface water from the ground beyond, and leave to deal with only the small amount of water that falls on the roast-yard itself. This water is best removed by tile drains, laid underground, with frequent openings at suitable places, where there is no danger of fine ore being washed into the drain.

They will, of course, have their discharge through the bank-wall into the ditch that runs between the lower track and the bank-wall. Assuming a fall of some ten feet between the spalling-shed and the ground under consideration, an elevated track is constructed over the central longitudinal axis of this rectangle, for the purpose of delivering the broken ore upon the heaps. Where no side-hill is available, the ore is carried up on to the heaps in wheelbarrows. The trestles to support the track may consist of sets or bents of two 8-inch by 12-inch posts with 8-inch by 10-inch caps six feet long. Bents 36 feet apart and properly braced. The posts should be about six feet apart at the bottom and two or three feet apart at the top.

These bents support the trussed beams, 10 inches by 12 inches, on edge, which carry the track as shown in the accompanying sketch. These girders may be made up of 2-inch or 3-inch planks spiked together.

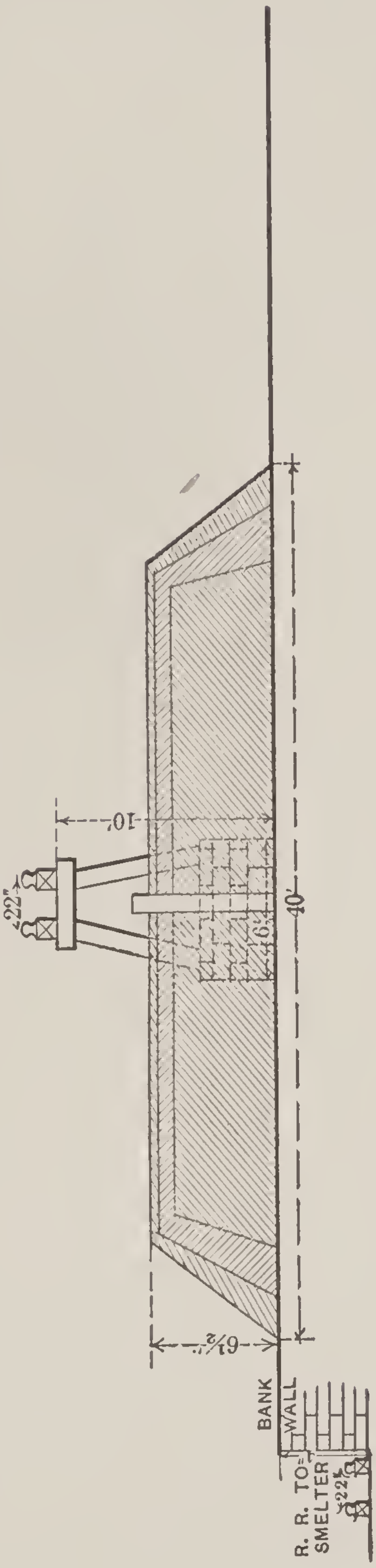
A fall of an inch in 12 feet will greatly facilitate the handling of the loaded car, and offer little obstruction to the return of the empty one. The track should, if possible, consist of T-rails, 12 pounds to the yard, firmly spiked to the longitudinal stringers, no sleepers being necessary; and well connected with each other by fish-plates, having two half-inch bolts at each end of each rail. All tracks throughout the entire establishment should have the same gauge; 22 inches is a convenient standard.

An iron-bodied end-dumping car, so made as to dump at right angles to the track, should be used. As the heaps are some 40 feet in length, the area over which the ore can be distributed by dumping from the car is far too contracted, and the following simple contrivance will be found to save many thousand dollars annually that would otherwise be expended in spreading the ore by hand: a plate of $\frac{3}{8}$-inch boiler iron, 30 inches square, fitted with a pair of short, low rails, on three sides of it, is so cut and



CROSS SECTION THROUGH ROAST HEAPS, PARALLEL WITH ELEVATED RAILWAY.

ELEVATION OF SAME PARALLEL WITH ELEVATED RAILWAY.



LONGITUDINAL SECTION THROUGH HEAP ACROSS RAILWAY.

placed upon the stationary track that the loaded car, striking first the flattened extremities of one set of the short rail pieces, while the flanges of the wheels run in corresponding slits until elevated upon the turn-table by the gradually increasing height of the short rails referred to, the heavy car may be easily turned upon the greased plate by a single workman, being held and guided to the similar pair of short rails placed at right angles to those already described by a circular guard rail, fastened at that end of the plate opposite to the point of entrance. A temporary track, formed of a pair of heavy rails, held firmly together, prevented from spreading by cross-ties, and supported by movable trestles, is laid at right angles to the main railroad, corresponding exactly to a pair of the short side rails on the turn-table plate. It will be readily seen that, by this simple contrivance, the extreme end of the longest roast-pile can be reached with the loaded car, while the turn-table plate can be shifted backward and forward until every square foot of the heap has received its proper quota of ore. The accompanying dimensioned drawing illustrates sufficiently the principal arrangements described in the preceding pages. If the contour of the surface permit, one longitudinal side of the prepared yard should be bounded by a wall about four feet in height, the top of the same being level with the ground on which the roast-heaps are built, while a railroad leading to the furnaces is constructed parallel with it, in such a manner that the calcined ore may be wheeled on a plank and dumped directly into cars without having to ascend any grade, thus greatly lessening the expense of loading. The labor and cost of preparing a plant, such as has been just described, will be quickly repaid by the consequent avoidance of the waste inseparable from a moist and muddy roasting yard, and especially from water flowing between the heaps. A case came under the author's observation, where the want of proper facilities for carrying off surface water caused a loss estimated at $12,000 within an hour, merely from the material washed away by the back-water from a swollen ditch, which passed between the roast-heaps, but which, from motives of economy, had been made too small to carry off unusual floods.

The height of the pile must depend entirely upon the character of the ore and the time for calcination at the disposal of the metallurgist. The higher the heap, the more fiercely it will heat, and the longer it will take to complete the operation. Consequently, where the ore is rich in sulphur, and when time is an object, as where the supply for the furnaces is small, heaps should be made low.

An ore with 12 per cent. sulphur, which is, perhaps, as low as can be thoroughly roasted in heaps without the intermixing of a considerable quantity of fuel throughout with the rock, may be piled up to a height of 7 feet advantageously, while solid pyrites with a sulphur tenor of from 35 to 40 per cent. should never be allowed to exceed 5 or 5½ feet, the measurement including only the ore, and not the layer of wood on which it rests. The best average height for ordinary ore is 6 feet, under which circumstances it will burn 70 days; the time being correspondingly diminished or increased by 10 days, if 6 inches be taken from, or added to, the above figures. The area of the heap has little influence on this time. The following table gives the result of the roasting of large quantities of various ores. In most of these cases, frequent sulphur assays were made of the ore under treatment; but in a few instances, the sulphur was estimated from a general knowledge of the material. The heaps were thoroughly covered and carefully watched, and the combustion was kept at the lowest point compatible with safety, the sole object being to obtain the most thorough possible roast, regardless of time or trouble.

This should be the universal practice; for although the grade of metal to be produced in the subsequent fusion may not demand such a thorough calcination, it is better to roast a certain portion of the stock thoroughly, and then reduce, or dilute, the matte to the required standard by the addition of raw ore. This lessens expenses in various ways. It costs but little more to roast an ore thoroughly than to do so partially; and the more completely the sulphur is eliminated from the roasted ore, the larger will be the proportion of raw ore that can be used in the charge; and consequently, the less will be the cost of calcining and the losses from fines of roasted ore. It is also very easy to keep the "pitch" or percentage of the matte produced at a proper point, when thoroughly oxidized stock is always at hand. These, and various other reasons that could be mentioned, are sufficient to refute the arguments of those who consider the addition of raw ore peculiarly injurious, and prefer an imperfect roasting of the entire stock.

LENGTH OF TIME CONSUMED IN BURNING HEAPS OF VARIOUS HEIGHTS.

Height in feet.	Quality of ore.	Per cent. sulphur.	Per cent. copper.	Days burning.	No. of sample.
5	Pyrite	39	6½	54	No. 1
5	Chalcopyrite, with little pyrite in quartz	18	14·3	41	“ 2
5	Bornite and pyrite	31	21·4	53	“ 3
5½	Same as No. 1	39	6½	66	“ 4
5½	“ “ No. 2	18	14·3	50	“ 5
5½	“ “ No. 3	31	21·4	65	“ 6
6	“ “ No. 1	39	6½	72	“ 7
6	“ “ No. 2	18*	14·3	61	“ 8
6	“ “ No. 3	31	21·4	74	“ 9
7	“ “ No. 1, much matted	39*	6½	94	“ 10
7	“ “ No. 3	31	21·4	86	“ 11
7½	Copper glance and pyrite in quartz	20*	23·4	54	“ 12

* Estimated.

The area of the heap is determined by the position and size of the ground at disposal, and the convenience of delivering the ore. Its width is limited by the distance to which the covering material can be conveniently thrown with a shovel, and by the room between the bents that support the track. 24 feet in width by 40 in length is a very convenient size, smaller heaps demanding considerably more labor and fuel to the ton of ore. With 36 feet between the bents, an ample border of 6 feet will be left on each side of the pile for collecting the fines, wheeling the same wherever required, and fully securing the wood-work against all danger of fire. Risk from fire is further obviated by elevating the foundation sill from which the uprights arise, upon a wall of slag-brick, 3 feet or more in height. A pile of the dimensions referred to, 24 feet by 40 feet square, and 6 feet high, will contain about 240 tons of ordinary ore, and should be built in the following manner:*

The corners of the rectangular space on which it is to be erected should be indicated by stakes, or, if the same size is to be permanently retained, by large stones, or better, blocks of slag, imbedded in the ground. The sides of the area being indicated by lines drawn on the ground to guide the workman, the entire space should be covered evenly to the depth of four or six inches with fine ore from the spalling-shed. This layer of sulphides answers several purposes: in the first place, it prevents the baking and adhering to the ground of the coarser ore, which, especially when much matte is formed, sticks to the clayey soil to such an extent as to tear up and injure the foundation, besides mixing worthless dirt with the ore, and causing a loss of the latter when attempts at separation are made. It also forms a distinct boundary line between the worthless and valuable materials, and, when left undisturbed during two or three operations, becomes itself so thoroughly desulphurized that the upper half or more may be scraped up with shovels and added to the roasted ore, its place being filled by a fresh supply of fines. This operation completed, the fuel is next arranged by an experienced workman in a regular and systematic manner. The quality and size of the wood is a matter of some moment, and must be determined for each individual case, it being evident that that variety of fuel that yields the greatest amount of heat for the longest time possesses the highest money value, provided the ore is of such a nature as to bear the temperature produced without fusing. As most sulphide ores will not stand the heat generated by a thick bed of sound, dry, hard wood, it frequently happens that a cheaper variety answers the purpose better. The outside border of wood that corresponds to the edges of the heap should be of better quality, as no such degree of heat is attainable there as in the interior of the pile. Therefore a large proportion of the bed may be made up of old rails, logs, gnarled and knotted trunks that have defied wedge and beetle, and such sticks of cord-wood as are daily thrown out from wood-burning boilers and calcining-furnaces as too crooked and misshapen to enter a contracted fire-place. Such miscellaneous fuel causes somewhat greater labor in arrangement; but whatever the material, it must be placed with such care and skill as to form a solid and sufficient bed, varying in depth from 8 to 14 inches, according to the behavior of the ore. However rough and irregular the greater portion of the fuel at our disposal may be, enough cord-wood of even length and diameter should be selected to form a four-foot border around the entire heap and just within the side-lines of the area; for the even and regular kindling of the heap depends considerably upon the proper arrangement of this border. Sticks of cord-wood not larger than 5 inches in diameter should be laid side by side across both ends and sides of the area. Across this layer, small wood is again piled until this four-foot border has been built up to the height of some 10 inches, brushwood and chips being scattered over the surface to fill up all interstices, while canals 6 inches wide, filled with kindlings, are formed at intervals of 8 or 10 feet, leading from the outer air and communicating with the chimneys in the center line of the heap. The empty area within this encircling border is now filled with the poorer quality of fuel, all sticks laid parallel and with as much regularity as possible, to cover all cracks and interstices, that no ore may fall through the wood, and to cover over the draught-canals in such a manner that they shall be neither choked nor destroyed by the superincumbent load.*

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* Subsequent experience has shown that it is more economical to still further increase the size of the heaps, 40 by 80 ft., and 7 ft. high is none too large.

The chimneys, which assist materially in rapidly and certainly kindling the entire heap, are formed of four worthless boards nailed lightly together in such a manner that two of the opposite sides stand some eight inches from the ground, thus leaving spaces that communicate with the draught-canals referred to, and toward which several of the latter converge. For a heap 40 feet in length, three such chimneys, eight inches square, will suffice. They should project at least two feet above the proposed upper surface of the structure, that no fragments of ore may accidentally enter the flue opening and destroy its draught. In certain localities, where even old boards are too valuable to be needlessly sacrificed, two or three medium-sized sticks of cord-wood may be wired together to form the chimney; or old pieces of sheet-iron, such as condemned jig-screens, worn-out corrugated roofing-iron, etc., may be so bent and wired as to form a permanent and sufficient passage, while this material will answer for several operations. The chimneys being placed in position, equidistant, and on the longitudinal center line of the bed of fuel, and held upright by temporary wooden supports, the heap is ready to receive the ore. This is brought in car-loads of 1,500 or 2,000 pounds from the spalling-shed, and weighed *en route* on track-scales. It is dumped on a portable wooden platform about eight feet square,

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*An excellent paper on heap roasting in Vermont, by Mr. William Glenn, may be found in the *Engineering and Mining Journal* for December 8, 1883.

to prevent the deranging of the wood from the fall of so heavy a mass of rock from a height of ten feet or thereabout. The first few car-loads are heaped about the chimneys, and the platform is changed from place to place as convenience demands, until the bed of wood is thoroughly protected by a thick layer of ore. The remainder of the process is a very simple operation. The cars of ore are dumped in turn over the entire area by a systematic shifting of the temporary pair of rails already described, and the heap formed into a shapely pyramid, with sharp corners and an angle of inclination of some 42 degrees, or as steep as the ore will naturally lie without rolling. The main body of the structure is formed of the coarsest class of ore; the ragging is next placed upon the pile, forming a comparatively thick covering at the part nearest the ground, and gradually thinning out toward the top and on the upper surface. Its thickness depends on the amount available, and no fears need be entertained of its having an unfavorable influence on the calcination; for when carefully separated from the finest class, a heap formed entirely of ragging will give reasonably good results. The extreme outside edge of the ore, when all is in place, should not entirely cover the external border of wood. At least a foot of uncovered fuel should project beyond the layer of ragging, both to prevent the ore from sliding off its bed as well as to insure a thorough kindling of the outer covering of mineral. The amount of wood required properly to burn a heap of 240 tons of ore will vary greatly with the composition of the latter, standing in direct proportion to its sulphur contents, and especially to the amount of bisulphides present, but may, on the average, be estimated at 12 cords, or one cord of wood to 20 tons of ore. In smaller heaps, this proportion must be considerably increased.

The fine ore that is to form the external layer, and on which depends largely the success of the process, is seldom placed upon the heap until after it is fired. Perhaps the most judicious practice is to cover the sides of the pile with a very thin layer, scattering it evenly with a shovel, and leaving the upper surface, as well as a space eighteen inches broad at the bottom, uncovered; for if the fine ore is thrown carelessly upon the lower circumference of the pile, the draught is decidedly hampered and the fire stifled before getting fairly under way. For an average ore, an amount of fines equal to 10 per cent. of its total weight is ample; of this, eight tons may be strewn lightly upon the sides of the heap, as just described, the remaining 16 tons—assuming the entire contents to be 240 tons—being arranged in small piles upon the empty space between the roast-heaps, where it is easily accessible to the shovel. The lighting should be done just as the day shift is quitting work, as the dense fumes of wood smoke, strongly saturated with pyroligneous acid and the various gaseous compounds of sulphur and arsenic, among which sulphureted hydrogen is always plainly distinguishable, are almost unbearable.

If possible, fine weather should be selected for this purpose; for although no ordinary rain is capable of extinguishing a well-lighted roast-heap, it may still interfere greatly with kindling a new one, and is quite likely to cause subsequent irregularities in the course of the process. There are several different methods of firing a roast-heap—such as lighting it only on the leeward side, and letting the fire creep back against the wind, kindling it through the draught-chimneys, etc., each of which has its advocates among roasting foremen; but long-continued observation has shown that no advantage is gained by any of these irregular methods, and the most sensible and successful practice is to light it as quickly and thoroughly as possible by applying a handful of cotton waste, saturated with coal oil, or a ladle of molten slag, to the kindling-wood at the mouth of each of the draught-canals, these being some six or eight in number, as already described. As the success of the entire operation depends principally on the management of the heap for the first few days after kindling, it will be necessary to study somewhat in detail the phenomena that it should normally exhibit during this critical period, always bearing in mind the impossibility of laying down any fixed rules that shall apply to all circumstances and to every variety of material

Under ordinary circumstances, the heap may best be left entirely to itself for from four to six hours after lighting, care merely being taken that the kindling burns freely, and that the draught-holes communicate with their respective chimneys. At the expiration of this time, if the fire has spread well over the entire area, about one-half of the remaining fines that have been provided for covering should be scattered lightly upon the heap; the lower border and upper surface, which have hitherto been left unprotected, now receive a thin application, while the lateral coating is rendered somewhat thicker and more impervious. If matters pursue a normal course, the early morning—twelve hours after firing—should see the heap smoking strongly and equally from innumerable interstices produced by the settling of the whole mass, due to the disappearance of the thick foundation of fuel. Dense pillars of opaque, yellow smoke, smelling strongly of sulphurous acid, arise from the site of each chimney; although if these were constructed of wood, no sign of them will remain except a few charred fragments, resting in a slight depression, which marks their sites. The entire surface will be found damp and sticky, and the covering material will have already formed quite a perceptible crust, from the adhesion of its particles. This "sweating," as it is termed, arises from the distillation products of the fuel—owing to its very imperfect combustion—and from the moisture contained in the ore. A yellowish crust surrounding the vents from which the strongest currents of gas are seen to issue indicates the presence of metallic sulphur, the volatilization of the first loosely bound atom of which begins soon after the wood is fairly lighted. Its quantity depends on the proportion of bisulphides in the roast, as well as on the freedom with which air is admitted; the scarcity of oxygen and a rather low degree of heat favoring its direct volatilization, while an abundance of air and a comparatively elevated temperature influence the plentiful generation of sulphurous acid.

During this first day, the newly kindled heap will require close and constant attention to prevent any undue local heating; nor is it at all uncommon to find that some neglected fissure has increased the draught to such an extent as to cause the sintering or partial fusion of several tons of ore at that point. The principal signs by which the experienced eye judges of the condition of affairs are the color of the gas and the rapidity with which it ascends; the amount of settling and consequent fissuring of the covering layer; and, above all, the degree of heat at different parts of the surface. A light, bluish gas, nearly transparent, and ascending in a rapid current, is a sign that the heat is too great at that point, and the admission of air too free. The fissuring of the crusted covering material, after the general and extensive sinking caused by the consumption of the fuel, indicates a rapid settling that can only arise from the melting together, and consequent contraction, of the lumps of ore. All these conditions are met by a single remedy; that is, covering the surface at that point more thoroughly with fines, by which means the air is excluded, the rapidity of the oxidation process diminished, and the temperature lowered. It should not be supposed that, because the interstices that exist in the upper part of the heap alone show evidences of heat and gas, those cracks and openings that have been left nearer the ground are of no importance; these are the draught-holes, while the former constitute the chimneys, and it is to the condition of the lower border of the pile that our attention should be most frequently directed in regulating the proper admission of air. A few shovelfuls of fine ore judiciously applied at the base of the heap will often have more effect than a car-load scattered aimlessly over the surface.

Only an experienced laborer can manage a roast-heap to the best advantage, nor is it possible to establish fixed rules for the guidance of this process, varying conditions demanding totally different treatment. In a general way, it may be said that, after somewhat subduing the intense heat caused by the sudden combustion of so large an amount of wood, the attendant should confine himself to scattering the covering material in a thin layer over the sides and top of the structure, and effectually stopping up such holes and crevices as seem to be the vents for some unusually heated spot below.

By the third day large quantities of sublimated sulphur will be found upon the surface, in many places melting and burning with a blue flame. It is now necessary for the attendant to ascend to the top of the heap, to properly examine the upper surface, and place additional covering material on such portions as still seem too hot. In doing this, a disagreeable obstacle is encountered in the clouds of sulphurous gas, which, to one unaccustomed to the task, seem absolutely stifling. By taking advantage of their momentary dispersion by currents of air, and retreating when they become too thick, no difficulty need be experienced in covering the upper surface of the heap as thoroughly and carefully as any other part of it.

If the process of combustion seems to have spread equally to all parts of the pile, nothing need now be done except daily to scatter a few shovelfuls of fines over such heated spots as seem to require it; but if any isolated corner of the heap has failed to kindle, or, having once caught fire, has now become cold and ceased to smoke, it is necessary to draw the fire in that direction. This can be accomplished with ease and certainty by any one accustomed to the work; for there is no danger of a roast-heap becoming extinguished when once fairly kindled. Certain isolated spots—especially corners and angles—may fail to become properly ignited, but by opening a few draught-holes in the neighborhood the fire will surely spread wherever unburned sulphides still exist. Beginning at the end of the first week, and continuing for a month or more, a certain amount of sulphur may be obtained by forming 18 or 20 circular, ladle-shaped holes about 14 inches in diameter and 7 inches deep in the upper surface of the heap, and lining them carefully with partially roasted fine ore, so that they may retain the molten metalloid. The impure sulphur may be ladled out twice a day into wooden molds; but the impurity of the product, caused by the great quantity of ore-dust and cinders constantly falling into the melted material, and the extremely scant production of a substance that is hardly worth saving, discourages the general adoption of the practice, although at some of the older German works it is still kept up. Experiments made with the greatest possible care saved only one-tenth of one per cent. of the total weight of the ore from a 30 per cent. bisulphide ore.

With certain varieties of ore, the sulphur, instead of collecting in a concentrated form at the principal issuing vents of the strongest currents of gases, condenses over the entire surface in a thin layer, and upon melting cements and agglutinates the fine particles of the covering layer in such a manner as to form an almost impermeable envelope. In such cases this crust must be destroyed, from time to time, with an iron garden-rake, or the process of calcination may be delayed for weeks beyond its customary limit from the lack of sufficient oxygen to maintain the proper rate of combustion. If arsenic is present, even in the smallest quantities, it will soon make itself visible as beautiful orange-colored realgar, AsS, and minute clusters of white, glistening crystals of arsenious oxide, which usually form at the upper orifices of the accidental draught-canals that communicate with the interior of the heap.

A strong and persistent wind from any one direction has an unfavorable effect on the process of heap-roasting, driving the fire toward the leeward side, and cooling those portions that feel the direct influence of the air-current to such an extent that one-fourth or more of the heap may remain in a raw condition. It is a somewhat remarkable fact that, while it is almost impossible to quench a roast-heap with water, unless completely flooded for a considerable length of time, a simple excess of the very element most favorable to its perfect combustion should have the power to extinguish it. If this annoying circumstance repeats itself with any frequency, it will be necessary to erect a high board fence on that side of the yard whence the most persistent winds prevail. Rain and snow have little influence on the course of the process, except in so far as they may cause serious chemical and mechanical losses. It is only after a heavy shower or sudden thaw that the great advantage of numerous and well-preserved ditches surrounding the entire area, and even leading between the heaps themselves, is fully realized and appreciated. When wet weather supervenes, after a long period of drought, the amount of copper dissolved from the soluble sulphate salts formed during the extended term of dryness may be so large as to repay some efforts to recover it. By simply leading the drainage from the roast-yard into two old brewer's vats partially filled with scrap-iron, during one summer, 3,546 pounds of 40 per cent. precipitate were collected.

During the last two-thirds of the life of the roast-heap it hardly requires an hour's labor, and if the works possess an ample stock of roasted ore in advance, nothing further need be done to the pile until it has burned itself out and becomes sufficiently cool to handle. The daily inspection, however, should never be omitted; for even at this advanced stage of the process, irregular settling or swelling of some portion of the structure may cause sufficient fissuring and consequent admission of air to produce serious matting, a disaster that the application of a single shovelful of fines at the beginning of the trouble would have prevented. In fact, it is far better to leave the heap undisturbed, unless good reasons exist for breaking into it, as the agglutinated covering material forms a roof almost impermeable to rain and wind, while the freshly calcined ore, when exposed to these elements, necessarily undergoes a serious waste. But if, as is in most instances the case, the demand for ore from the smelting department exceeds the supply from the mine, but scant time can be afforded to the intermediate steps, and the calcination must suffer. If, therefore, it is the object to utilize, at the earliest possible moment, the ore that is stored up in the heaps, they should be closely watched, and whatever portions of the same—usually the ends and corners—are found to be moderately cool, should be carefully stripped and broken into, the object being to cool the ore that is already roasted, and extinguish the last remains of fire as rapidly as possible, without interfering too seriously with the process of oxidation that is continuing in the main body of the pile. This is accomplished by digging away the calcined ore, and following up the line of fire as it recedes from the surface toward the center, without approaching it so closely as to completely extinguish it in that portion of the ore not yet properly calcined, which is easily done at this stage of the operation. At least 12 inches should be left between the outer air and the line of active oxidation, and it is a good practical rule never to allow the surface to become so hot as to be unbearable to the naked hand.

The too common practice of keeping the smelting department so far in advance of the ore supply as to require the breaking into and utilization of roast-heaps in which the ore is still red-hot, and just at the most active and profitable stage of calcination, necessitates the employment of a strong body of laborers to bring water and constantly drench the smoking ore, in order to make it at all possible for the other workmen to shovel it into their barrows, and must be condemned as unnecessary and productive of more trouble and expense than almost any other practice at our smelting-works.

Among these sources of extra expense are the doubled cost of taking down and transporting the roasted material; the burning and rapid destruction of tools and cars; the medical bills claimed by the workmen who suffer from such unhealthy employment; and, far greater than all, the injurious effect on all subsequent steps of the process, which will be referred to in the chapter on Smelting in Blast-Furnaces.

On the other hand, the only possible advantage that can be claimed is, that some two or three weeks' interest on the value of the ore is saved.

When the heap is properly cooled, the mass of ore, which, while still hot, is often almost as hard and tough as a wall of solid rock, crumbles to pieces with a single blow of the pick.

and is wheeled in barrows from the roast-heap to the furnace-car.

When the heap is sufficiently cooled, it is "stripped" by removing not only the fines that formed its cover, but its entire surface to such a depth as is necessary to include all material that has escaped oxidation. This unroasted material is made up largely of the fines forming the cover, and which, though often quite thoroughly oxidized on the top of the pile, are so agglutinated with sulphur as to be unfit for the furnace. The covering of the sides is seldom sufficiently roasted, and this is especially the case near the ground, where the ragging itself, to a depth of several inches, is frequently found unscathed. The angles of the pile are also seldom in good condition, and many isolated patches and bunches of ore will be found that the careful foreman will reject. This statement, however, refers rather to the results of the ordinary practice than to those that can easily be obtained by close attention to details and by enlisting the interest of some intelligent foreman. As already explained, the fire will find its way to every nook and corner where sulphides still exist, if only the conditions are favorable. The author recollects with satisfaction the mortification displayed by his roasting foreman but a few months ago, at the unusual occurrence of a few hundred-weight of fused, and a still smaller amount of raw, ore in a heap of some 200 tons.

A half-fused, honey-combed condition of the upper part of the heap, presenting the appearance of a skeleton of gangue from which all mineral has been melted out, is a certain indication of a proportional amount of matte below. This molten material naturally gravitates to the bottom of the heap, and is there found in masses of greater or less extent; often of many tons' weight, though, in such a case, warning would have been given during the roasting by the irregular sinking of the heap, and even by depressions and crater-like cavities on the surface. This molten product is very properly termed "heap-matte," and varies neither in appearance nor composition from the similar product of a blast-furnace. A popular impression prevails among certain foremen, and even assayers, that the light honey-combed material that remains after the melting out of its sulphide constituents is rich in copper, but the contrary is true. The unfused skeleton merely represents the siliceous slag, while the molten sulphide mass below is the equivalent of the matte, the purity and value of either product depending on the temperature to which the ore has been subjected, and the consequent perfection of the smelting or liquidation process. This fact is sustained by the following assays of samples of considerable size:

	No. 1.	No. 2.
Original ore before roasting......	21·6 copper.	18·6 copper.
Siliceous skeleton...............	7·3 "	6·4 "
Heap-matte.....................	34·7 "	36·6 "

The formation of this heap-matte in any considerable quantity is very detrimental to the roasting process, but is easily avoidable; for it is invariably caused by either too much or too little air. In too many instances, no particular notice is taken of its occurrence, and it is sent to the smelting-furnace mixed with the well-roasted ore. This is exceedingly bad practice, and should on no account be permitted, as it is totally impossible to foresee the grade of matte that will be produced by the smelting process when this unroasted sulphide is mixed in unknown and varying quantities with the properly prepared charge. If the percentage of the furnace mixture be such that the addition of this raw matte does not lower the tenor of the product below the desired standard, it may then, of course, be fed with the roasted ore, but should be kept strictly by itself, and added to each charge in weighed quantities. Any infringement of this rule gives rise to the formation of a matte varying greatly in its percentage of copper as well as in its entire composition, and deranges not only the smelting process, but seriously affects the regularity of the matte concentration operations.

The heap-matte may occur in such masses that serious difficulty is experienced in breaking it up, especially as it retains its heat for a great length of time, and in this condition is almost malleable, yielding and flattening under the blows of the sledge like a block of wrought-iron. Much expense and annoyance may be spared by stripping the central molten mass thoroughly of all adhering ore, and allowing it to cool for two or three days; at the expiration of which time it will be found quite brittle and comparatively easy to deal with. Thorough and repeated drenchings with water will produce even better results; but it should be borne in mind that a considerable proportion of the cupriferous contents of calcined ore is in a soluble condition.

When, through carelessness or inexperience, heap-matte is formed, it must be either treated together with the matte produced from the first fusion in the blast-furnace, or set aside until a sufficient amount is collected to form a small heap by itself, and be re-roasted. It should, on no account, be mixed with the raw ore, as it demands a different treatment, and will either cause irregularities in the ore-roasting, or will pass through that process unaltered and with no perceptible diminution in its percentage of sulphur.

The proportion of strippings and other unfinished products of heap-roasting that may be considered allowable was determined experimentally by simply weighing the finished and unfinished portions of half a dozen consecutive roast-heaps, averaging about 240 tons each. About 10 per cent. of fines were used for the covering layer in each case. The total amount of unroasted material, as given in the following table, shows that even a portion of the fines is thoroughly oxidized:

	Unroasted. Per cent.	Roasted. Per cent.	Days heap was active.
No. 1..................	9·6	90·4	64
" 2..................	6·6	93·4	71
" 3..................	8·4	91·6	70
" 4..................	9·0	91·0	61
" 5..................	7·6	92·4	67
" 6..................	11·4	88·6	57

The figures have been slightly corrected, without altering their relative values, to make the aggregate in each case exactly equal 100 per cent., which, of course, can never be precisely attained by adding the weights as actually arrived at.

While these results are taken from ordinary every-day work, it should be understood that they can only be attained by the most careful attention in the roasting-yard. The proportion of the product rejected as unfit for the smelting-furnace at some works might be even less than in the cases just cited, and the reason may be readily recognized in the low grade of the product from the fusion, and the constant complaints of the impossibility of keeping the matte up to the proper standard. A selection in such cases as rigid and thorough as in those just tabulated would result in the rejection of from 25 to 60 per cent. of the entire heap. An allowance of 10 per cent. may therefore be considered reasonable—although demanding more than ordinary care and skill—and of this three-fourths should be fines. The stripping should be performed in a cleanly and systematic manner, and to an extent several feet in advance of the line of excavation, and the material thus removed piled on one side, to be subsequently screened on the first calm day; for the least wind causes a heavy loss when handling this half-oxidized powder. The fine part is again used as a covering, for which it is much better suited than raw ore, while the much smaller coarse portion is added to the nearest heap in process of erection.

It will be readily seen that very much more fine ore is produced during the processes of mining and crushing than can be used for the purpose of covering material, especially as only a small proportion of the latter is sufficiently oxidized at each operation to be passed on to the smelting-furnace. The problem of the best means of utilizing this constantly increasing amount of fine ore in works unprovided with calcining-furnaces is often a pressing one. It will be referred to again, under the heading, "The Treatment of Pulverized Ores."

The roast-heap, when once tolerably cool, is torn down and loaded into the furnace-car with great celerity. Three or four men trundle the barrows, while double that number wield the pick, shovel, and hammer. It is the duty of these laborers to break all partially fused masses or lumps that are too large for proper smelting into fragments of a reasonable size, as especially determined by the metallurgists. There is not time, or space, or opportunity on the charging-floor of a blast-furnace in full operation to attend to any duties beyond those immediately connected with weighing the charge and filling the furnace, and many serious irregularities in the smelting may be traced to an omission of this simple and obvious precaution.

A careful and humane foreman can do much to mitigate the annoyance and suffering to which the workmen are subjected during the labor of tearing down a heap, by moving the point of attack from one to the other side of the pile, according to the direction of the wind, as well as by keeping the fresh surface on which the men are engaged well sprinkled with water, to settle the fine ore-dust. At best, this labor is the most disagreeable and wearing connected with ordinary smelting, and no laborer should be kept at such employment for more than three or four days in the week, and should be changed to some other task during the remaining time. Aside from the common tools already enumerated, long, stout steel gads and a few heavy sledges are needed to break up the central portion of the structure, which, although not fairly fused, is often so stuck together as to require considerable labor for its removal. At no other work are shovels so rapidly destroyed, and it is to this place that all partially worn, though still serviceable, tools are sent to terminate their existence.

The tearing-down of the heap, and breaking-up of the matte that may be formed in it, are greatly facilitated by the use of a small quantity of dynamite, or other high explosives, selecting a powder of rather low force; containing not over 30 per cent. of nitro-glycerine.

When this is properly used and in not too large quantities, it saves infinite labor with bar and pick, a single shot placed in a hole made in half a moment's time with a bar, often accomplishing more than hours of hard labor. The shot should simply shake up and loosen the mass, leaving the large lumps to be broken up by sledge and pick, as usual. If enough powder is used to break the whole mass up into small fragments, a great portion of the ore will soar into the air and go toward top-dressing the surrounding country.

I have never been able to get my men to be economical enough with their powder, excepting by forcing them to pay for it themselves. When they realize that every penny that is saved on powder goes into their own pockets, it is astonishing how little it takes to do the same work that required several times the quantity when it cost them nothing.

After the complete removal of the old heap and any slight repairs that may be required to restore the ground to its former level, a thin layer of raw fines is again spread on the old spot, and the fuel arranged for a fresh pile. The estimate of costs for this process, as given below, is based on the treatment of a very large amount of ores, varying greatly in composition, and under very various circumstances, and is purposely made somewhat liberal to allow for the occurrence of bad work and various other mishaps that are certain to occur in a greater or less degree. It is based upon a plant of 100 tons' daily capacity, and on the assumption of only a short distance for transportation of the roasted ore to the smelting-furnace.

ESTIMATE, 100 TONS PER 24 HOURS.

Transportation to heaps, 2 men on car, weighing 1 man—3 men, at $1.50	$4.50
Labor—building and burning heaps— 4 men, at $1.50 = $6.00 2 men, at $2.00 = $4.00	10.00
Fuel at the rate of 5 cords per 24 hours at $5 a cord	25.00
Removing and loading roasted ore, 14 men at $1.50	21.00
1 foreman	2.50
Transportation to furnace and weighing (same as to heap)	4.50
Oil, repairs to cars, track, etc.	3.25
Miscellaneous labor, screening, daily patching, etc., 2 men at $1.50.	3.00
Renewing shovels and other tools	4.00
Repairs on gads, bars, and tools	2.75
Total	$80.50

Or 80½ cents a ton of raw ore. On deducting the cost of the double transportation, as well as that portion of the labor belonging to the loading of the cars for the smelter, and for repairs to cars and track, etc.—none of which expenses actually belong to the process of heap-roasting as often estimated—the entire cost is at once reduced to 50 cents a ton, or thereabout.

The various operations of heap-roasting may often be performed by contract to great advantage, especially if one has a good foreman to see that the quality of the roast is kept up to a satisfactory standard.

To give an idea of the prices that are fair for this operation, I will mention what I paid for roasting a heavy, pyrrhotite ore in large quantities, say 150 to 200 tons per day, the climate being excessively cold and stormy, and laborers' wages about $1.40 per 10 hours. The company furnished the wood for the roast-beds, and delivered the cars at the yard; but the cars had to be unloaded by hand and the raw ore wheeled to the heaps, the arrangements for dumping the ore direct not having been then completed.

For unloading the raw ore on the heaps, laying the wood, completing heaps, and covering and watching them throughout the entire operation, $0.22 per ton of ore.

For stripping, tearing down, and loading the roasted ore on cars, and unloading the cars by hand into the smelter-bins, $0.16 per ton of ore.

The contractors furnished their own powder, but the company provided tools, barrows, etc., though the contractors paid for the sharpening of their bars, picks, etc.

On the above basis, the contractors made a fair profit when they attended strictly to their business, and when there were no interruptions or shut-downs.

The degree of desulphurization arrived at by this process is seldom accurately determined, owing to the difficulty and expense of obtaining an accurate sample, and to the fact that the experienced eye can very correctly judge of the success of the roast, while any defect in the process will become immediately apparent in the lower tenor of the product of the succeeding fusion. Owing to the scarcity of accurate investigations on the subject, the following determinations were made:

No. 1. A heavy pyritous ore, from the Ely mine, Vermont, consisting principally of magnetic pyrites and chalcopyrite, burned in a heap of about 300 tons for eleven weeks. After stripping off the surface, a sample of the roasted ore, as delivered at the smelting-furnace, was taken. The following was the assay of the ore before and after calcination:

	Before roasting.	After roasting.
Sulphur	32·6 per cent.	7·4 per cent.
Copper	8·2 "	9·1 "
Insoluble	27·0 "	31·1 "

The condition of the copper in the roasted sample was also determined in this case, as follows:

Sulphate of copper	1·3 per cent.
Oxide of copper	2·1 "
Sulphide of copper	5·7 "
Total	9·1 "

No. 2. A heavy pyritous ore, being almost pure iron pyrites containing minute quantities of copper, silver, and gold, from the Phillips mine, Buckskin, Colorado, was roasted for 6 weeks in piles of 60 tons, and was used as a flux for siliceous silver ores. A careful sample of the roast yielded sulphur, before roasting, 46½ per cent.; after roasting, 11 per cent.

A considerable number of similar tests give corresponding results, showing that a very fair degree of desulphurization can be attained by this crude and ancient method, but still better results will be reached in ores containing less pyrites, and making the fact evident that, in heap-roasting as well as in the calcination of pulverized sulphides, the copper is the last metal present to part with its sulphur, and that a large proportion of this still remains in the condition of a sulphide after nearly the entire iron contents have become thoroughly oxidized. This agrees perfectly with all investigations relative to the comparative affinity of sulphur for the various metals, and is in no other metallurgical process more strikingly exemplified than in the so-called "kernel-roasting," as practiced at Agordo, in Italy. There, the mechanical separation of the copper from its accompanying pyritous gangue is effected by stopping the process of calcination at the exact point where the entire iron contents have been oxidized into a soft earthy material, while the copper remains in combination with sulphur in a hard, metallic condition, and, most singularly, retreats into the center of each lump of ore, forming a heavy and solid kernel, which can easily be separated from its earthy envelope by inexpensive mechanical means. As this interesting process is not practiced in this country, and in all probability is not suited to our domestic conditions, the student desirous of pursuing the subject will find in Plattner's *Röstprocesse*, as well as in a paper by the author in the *Mineral Resources of the United States* (A. Williams, Jr., 1883), further information.

The appearance of a freshly-opened heap of well-roasted ore is characteristic, although difficult of description. It should present a strictly earthy, irregular surface of a blackish-brown hue, the scarcity of air preventing the oxidation of the iron to the red sesquioxide. This is a decided advantage in a reverberatory smelting-furnace, where the powerful carbonic oxide atmosphere of the blast-furnace is wanting, to reduce it to the protoxide, and thus fit it for entering the slag, the higher oxide being infusible at ordinary smelting temperatures. It is, in fact, principally a magnetic oxide, and, while the greater part of the contents should adhere closely together, and, when disturbed, should come out in the shape of large lumps, no sign of actual fusion should be visible, and the largest mass should fall into fragments at a few blows of the hammer. The more siliceous pieces of ore will have taken on a somewhat milky and opaque look in place of the ordinary vitreous appearance of quartzose minerals, and the veinlets of sulphides traversing the same will be found oxidized throughout. The solid lumps of pyrites, if carefully broken, will usually display a series of concentric layers, completely oxidized and earthy on the outside, and gradually acquiring greater firmness and a slight sub-metallic luster, which culminates in a rich kernel near the center of the fragment. This resembles strongly one or other of the grades of matte as produced from the smelting-furnace, and usually contains the greater part of the entire copper contents of the lump. The silver—if any be present—is also concentrated in a marked degree, though, so far as the author's own investigations extend, not with the same remarkable perfection as the less precious metal. The examination of a characteristic lump, such as just described, which contained before roasting about 4 per cent. of copper, yielded the following interesting results:

The outer earthy envelope contained..........	Traces		of copper.
The medium concentric layers.................	1·2	per cent.	"
The central sub-metallic kernel................	69·6	"	"

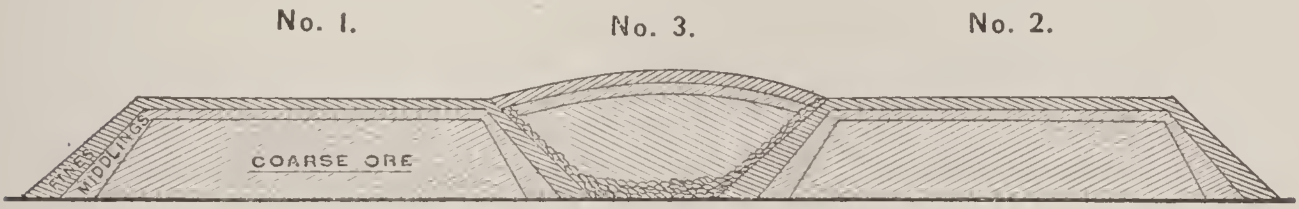
An imperfect roasting is quickly detected by the presence of more or less fused material at certain portions of the heap, while elsewhere there exists no cohesion between the lumps of ore, which fall apart like so many paving-stones. A certain metallic appearance will also be noticed, very different from the dull, earthy character of the properly burned pile. Although a large proportion of the contents may exhibit quite a brilliant red color, as though an unusually perfect oxidation of the iron had taken place, a mere weighing of one of the lumps in the hand will quickly undeceive the least experienced observer, and its fracture will show that the effect of the fire was only surface deep, while the entire interior remains unaltered. A careful study of different roast-heaps, wherever opportunity offers, will soon render the student skillful in judging by eye of the degree of success attained by this process, and in after-life frequently furnish him the key to the cause of the unsatisfactory tenor of the matte produced from his furnaces. No metallurgical process is more dependent upon an efficient and conscientious foreman, and the best results are usually obtained by selecting some intelligent and ambitious man from the roast-yard laborers, and holding him strictly responsible for results.

A decided improvement in heap-roasting of ores was introduced at the works of The Canadian Copper Company of Sudbury, Ontario, under the management of the author, in 1888–89.

It was first tried by his assistant, Mr. James McArthur, and proved so valuable that it became a regular practice under ordinary circumstances.

We have called it the "V-Method" of roasting, and the accompanying sketch will make it clear. It consists in introducing a supplementary roast-heap between each two regular heaps, so that, if left untouched, there would be a continuous and unbroken roast-heap the entire length of the roast-yard.

The supplementary heap should not be built until its two neighbors, which are to form its lateral walls, are well under way, and have been lighted from 10 to 14 days. By this time, if properly managed, they will be cool enough on the outside to run no risk of setting afire the bed of wood which is laid down for the supplementary heap. The fresh bed of wood is laid down rather thinner than for independent heaps, and a single layer is extended well up the slope of the two neighboring heaps. The ore is dumped on as rapidly as possible, and the heap finished off with ragging and fines in the usual manner, and fired from the ends.



THE V-METHOD OF HEAP-ROASTING.

The result is excellent, for the new heap, having its sides protected, burns clear through its entire extent, and then sets on fire the still unroasted ore on the outside of the two neighboring heaps.

Thus the proportion of unroasted ore is reduced to a minimum, and indeed is seldom worth keeping separate.

Another great advantage is the economizing of space, for by this arrangement, some 60 per cent. is added to the capacity of the roast-ground.

It may require some little patience and experimentation at first to adapt this practice to a new ore, but it is well worth the trouble, and has been pronounced by the leading members of our profession, a decided and important improvement in this ancient and useful process.

In the case referred to, the ore that was roasted was a nickeliferous pyrrhotite mixed with chalcopyrite; but I have tried it sufficiently on both heavy and lean ores of the ordinary yellow iron pyrites to know that it is equally well adapted to all ores that are any way suited to heap-roasting.

HEAP-ROASTING OF MATTE.

There remains only, in connection with this portion of the subject, to notice the slight deviations that it is found necessary to introduce in adapting this method to the treatment of matte.

These artificially formed sulphides, containing variable percentages of sulphur, may be sufficiently desulphurized in heaps, and their chemical composition has no marked effect upon the result, provided lead is not present to such an extent—fifteen per cent. or more—as to increase the fusibility of the material.

The most marked distinction between the behavior of ore and matte, when submitted to this process, is the fact that, while the former substance may be satisfactorily oxidized by a single treatment, the latter invariably demands two, and oftener three or more separate burnings before it is properly prepared for the succeeding fusion. There is no exception to this rule, which, if properly understood, would prevent the disappointment frequently experienced by those unaccustomed to this method of desulphurizing matte, and who are led to condemn the practice on finding, at the conclusion of the first carefully conducted burning, that the only visible results are a slight scorching of the surface of each fragment, a change in color from the original brownish-black to a brassy yellow, and a more or less extended fusion of such portions of the heap as have sustained the greatest heat. In reality, the influence of the process has been much more profound than can be realized from external appearances, and although neither the removal of the sulphur nor the oxidation of the iron and copper has progressed to any great extent, a certain change in the physical condition of every fragment of matte has been effected that prepares it perfectly for a second burning, and which seems a necessary preliminary to the actual desulphurization.

Each succeeding operation requires a slightly increased proportion of fuel, as the volatilization of the sulphur and the oxidation of the metallic constituents deprive the matte of its internal sources of heat, and at the same time greatly lessen its fusibility.

For the first roasting, a bed of wood should be prepared similar to that for a heap of ore, although smaller in area; for it is difficult to regulate the temperature and prevent matting in a heap much larger than twelve feet square, and this will be found a convenient size to hold from sixty to seventy tons of matte when raised to a height of about six feet. A single chimney in the center is sufficient, and about this structure the broken matte should be heaped just as it comes from the crusher or spalling-floor, and regardless of the fines that it contains. The presence of these has been found necessary to check the rapidity of the operation, and prevent the fire from suddenly spreading through the entire pile in a few hours without accomplishing any useful result, though generating for a short time a temperature high enough to fuse a large proportion of the contents into a single lump.

Less care need be taken in shaping a matte-heap than in the case of ore, and it is merely necessary to build it up in the form of a rude mound, which may best be covered with thoroughly burned ore from the roast-heaps, most of which on handling will crumble to a sufficient fineness for the purpose, while any hard lumps may be removed with the dung-fork. This obviates any screening or classifying of the matte in the open air, which always entails a heavy loss, owing to the great value and excessive friability and lightness of the material after calcination. If, as is usually the case, the proportion of fines after the first burning is found so great as to endanger the proper combustion of the heap for the second operation, the mechanical loss may be reduced to a minimum by separating the excess of pulverized matte by the use of a dung-fork, with tines closely set, during the turning of the ore from the heap just finished on to the fresh bed of wood, and at the conclusion of the process, removing the fines that are thus isolated, either directly to the smelting-house, or, if they still contain too much sulphur, to the calcining-furnaces. The covering of the original heap, consisting solely of roasted ore, should be stripped off, and either sent to the smelting-furnace or again used for a similar purpose. It need hardly be mentioned that the presence of arsenic or similar impurities in the ore, in greater quantities than in the matte, should prevent any such practice as that just recommended, and it may be accepted as a universal rule in copper smelting, that no impure ores or products should ever be mixed with those freer from deleterious substances.

Under no circumstances need a matte-pile be covered as thoroughly as a roast-heap consisting of ore, nor can the formation of a considerable amount of matte, which in ore-roasting would be evidence of a great want of skill or care, be considered as a reproach, experience having so conclusively shown the impossibility of preventing its occurrence that, unless about $\frac{1}{8}$ of the lower portion of a matte-heap is thus fused, no thorough oxidation of the remainder will be effected. The time necessary for the operations just discussed varies according to the quality of the matte, the condition of the weather, and certain other factors, but will in general be, for the first burning, eight days, while on the tenth day the heap will be sufficiently cool to permit its turning on to a fresh layer of fuel. The second operation requires a day longer, and the third a day less than the first burning.

To those familiar with the practice of heap-roasting as applied to ores, no particular directions are necessary except that care should be taken that the large blocks of matte that are formed during each burning be well broken up and placed near the center of the heap next constructed, that they may have every opportunity for a thorough desulphurization. Whatever raw matte still remains from the last burning is best reserved until the construction of a fresh heap furnishes the proper means for its treatment. At the last two burnings, it is well to introduce two or more layers of chips, bark, or other refuse fuel into the matte-heap; for it will act powerfully in decomposing the sulphates that at this stage are formed in considerable amount, and also exercise a similar and most marked effect on whatever compounds of arsenic and antimony may be present. This simple measure had a sufficient effect in a certain instance in the experience of the author to be plainly noticeable in the quality of the ingot copper produced.

No attempt to select such portions of thoroughly calcined material as will be found after the second burning has ever proved remunerative. The heap of matte must be treated as a whole, and the roastings continued until the desired grade of desulphurization is reached.

The process just described is seldom an advantageous one, as, aside from the production of the vilest fumes known to metallurgy, the value of the material operated on is too great to admit of being locked up for 30 days or more, or to warrant the loss that necessarily results from such frequent handling in the open air. The last difficulty may be partially obviated by erecting a light structure to protect the heaps from the rain and wind; but, at best, the practice is an imperfect and objectionable one, and only to be recommended in new, outlying districts, where an expensive calcining plant cannot at once be erected, and where the climate is favorable for out-of-door operations. The expense of crushing and calcining in furnaces is scarcely greater than the three or four burnings necessary to produce the same result; but the condition of the roasted material is so much more favorable for the succeeding smelting process, in the case of heap-roasting, that this reason alone is often sufficient to outweigh all objections that can be offered.

The practice of spalling the large pieces of matte upon the heap itself must be deprecated, as it has a strong tendency to solidify the structure and render the draught weak and irregular.

The cost of this process, based upon the roasting of many thousand tons of matte, and divested of those details that too closely resemble the heap-roasting of ore to warrant repetition, is as follows, assuming the daily amount of fresh matte subjected to this treatment to average 30 tons:

COST PER TON OF MATTE.

First Fire.	
Breaking	$0.35
Transportation to heap	0.15
Fuel—allowing 3 cords of wood to 60 tons of matte	0.25
Constructing heap and burning	0.32
Total	$1.07
Second Fire.	
Fuel—same as before with addition of chips	$0.30
Turning heap and burning	0.40
Total	$0.70
Third Fire.	
Fuel—same as second fire	$0.30
Removing finished heap	0.40
Transportation to furnace and expense of preparing the raw matte still remaining, which results from the fused matte	0.45
Total	$1.15
Total cost of 3 burnings	$2.92

CHAPTER V.

STALL-ROASTING.

At just what period in the history of the art it became customary to inclose the roast-heap with a little wall of earth or mason-work, in order to protect it against the elements, to concentrate the heat, and to render unnecessary the tedious labor of covering the sides with fine ore, is unknown, though Agricola's work on metallurgy shows that it was no novelty in the sixteenth century. These simple walls have since been heightened and sometimes connected with an arched roof; the area that they inclose has been paved and occasionally furnished with a permanent grate; and, more important than all, the interior of the stall has been connected by a flue with a tall chimney, by which the draught has been improved, thus shortening the process of oxidation, while the noxious fumes are discharged into the atmosphere at such a height as to render them unobjectionable in most cases.

A very great variation exists in the size, shape, and general arrangement of stalls, hardly two metallurgical establishments building them after the same pattern, though all essential differences may be properly considered by dividing them into two classes:

1. Open stalls, suitable only for ore.
2. Covered stalls, suitable for both ore and matte.

1. *Open Stalls.*—Any attempt at an exhaustive description of the different patterns of ore-stalls that human ignorance, as well as ingenuity, has invented, would be a waste of space. They all consist of a comparatively small paved area, surrounded by at least three permanent walls, and usually having an open front, which is loosely built up at each operation, to confine the contents. The back or sides, or both, are pierced with small openings communicating with a flue common to a large number of stalls that enters a high stack. The draught is confined to these

passages by covering the surface of the ore with a layer of fines. From the great variety of existing patterns, one built at the works of the Parrot Copper and Silver Company, of Butte City, Montana, is selected for description as possessing exceptional advantages as regards cheapness of construction, convenience of filling and emptying, economy of fuel, and general adaptability.

The stalls may be built either of common red brick, of stone, or, far better, of slag molded into large blocks, which, from their size and weight, require little or no extraneous support; while brick demands thorough and extensive tying together with ironwork, and stone of proper size and shape is expensive and is apt to crack when exposed to great fluctuations of temperature.

As these so-called "slag-bricks" are invaluable for walls and foundations, and in fact for every purpose for which the most expensive cut granite would prove available, and as they can be produced from almost any ordinary copper slag, a brief description of the cheapest and best method of manufacturing them is appended.

MANUFACTURE OF SLAG-BRICK.

These are generally made from the slag of reverberatory smelting-furnaces, both because this material is usually more siliceous than any other, and also because, during the process of skimming, it can be obtained in large quantities in a very brief space of time. There should be no difficulty, however, in making the brick from the slag of a blast-furnace, provided the smelting is sufficiently rapid to fill the molds properly, and that it is not so basic as to yield too fragile a material on cooling. Even with exceedingly brittle blocks, produced from a highly ferruginous ore, excellent and durable walls can be constructed, provided the blocks are placed in position uninjured; for they will bear an immense crushing weight with impunity, and seem to defy the action of the elements.

Assuming the slag to be obtained from a reverberatory furnace, the process of preparing the molds should be begun as soon as possible after the slabs from the previous skimming have been removed and all chips and fragments cleared from the sand bed by the aid of a close-toothed iron garden-rake. Ordinary loam—or a natural mixture of fine sand and clay of such consistence that, when slightly moistened, it will ball firmly in the hand—is the proper material for the molds, which should be formed by means of a number of wooden blocks, of the required size, carefully smoothed and slightly tapered to facilitate their removal from the sand, and furnished with a 30-inch handle, inserted in their upper surface. These slag blocks are molded on the flat, in the same manner as ordinary red brick; and after leveling off the pile of dampened sand to form a smooth and horizontal bed, the wooden blocks—some twelve in number on each side of the skimming door—are arranged in a double row, four inches apart between blocks, and the same distance between the two parallel rows.

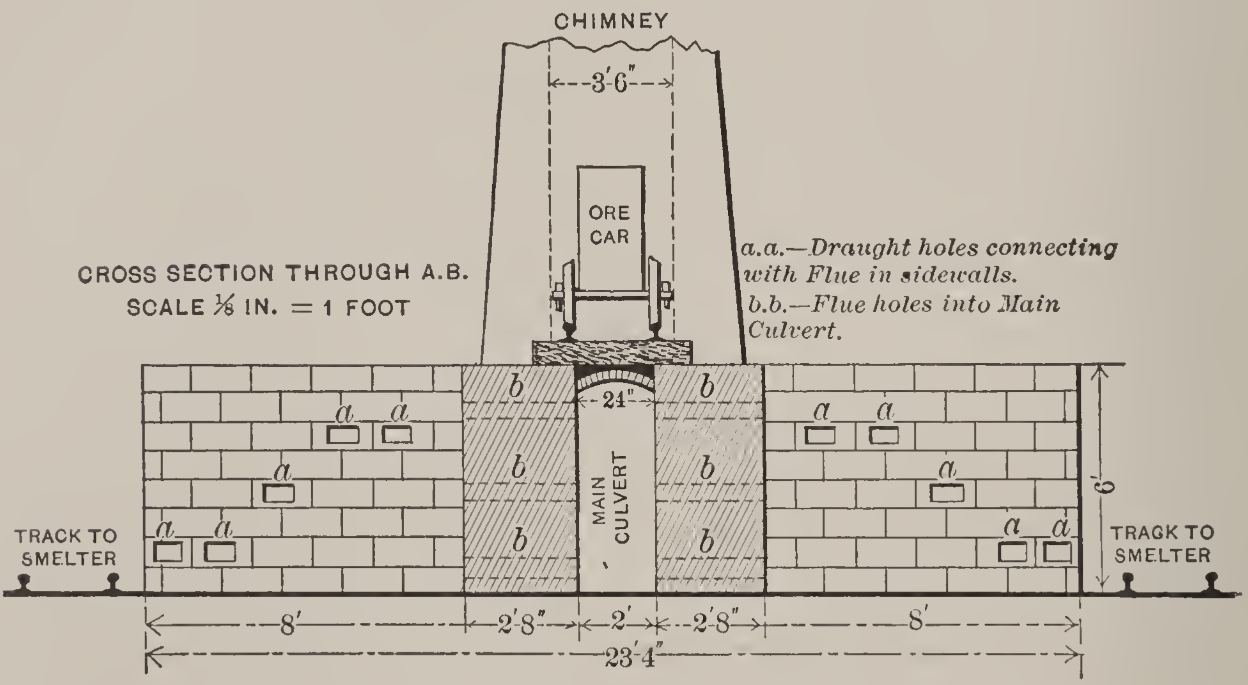
Besides the ordinary deep excavation for the plate slag, a second bed should be left on each side, between the former and the first brick mold right and left, both for the purpose of settling any grains of metal that may be accidentally drawn over during the process of skimming, and to act as a regulating reservoir to lessen the sudden impulse of the waves of slag that follow each motion of the rabble, and thus to prevent the destruction of the very fragile sand molds. The entire bed is constructed on an inclination of about one-half inch to the foot; the plate slag forming the summit, while the double row of molds slopes away from it in each direction laterally. After the wooden blocks have been placed on this sloping bed in a proper horizontal position, and exactly equidistant from each other, as determined by a wooden gauge, the remaining sand, very slightly but equably dampened, is shoveled back again, and carefully trodden and tamped evenly into all the interspaces and around the outside edges of the blocks, until it reaches the level of their upper surface. This is a very brief operation; for it is not essential to tamp the sand very firmly so long as about an equal degree of solidity is imparted to all portions of it. A cylinder of hard wood—three inches in diameter and four inches long—which, when placed lengthwise, fits exactly between each two molds, is laid upon its side, and, by a few blows of the mallet, driven into the sand, thus when removed forming a little gutter through the middle of the partition wall, and connecting each pair of adjacent cavities in such a manner that the flow of slag through either entire lateral system meets with no impediment. The wooden blocks are then removed from their sand bed with the greatest care, it often being necessary to loosen them by gentle tapping and other means familiar to the experienced molder. The bed requires only a few hours' drying to fit it for the slag.

By the time the charge is ready for skimming, say in three hours or less after the completion of the bed just described, it should be in proper condition, and the furnace helper, armed with a small rabble-shaped hoe, stands beside the skimmer ready to turn the stream of slag into the proper molds, remove obstructions from the gutters, break through the rapidly forming crust if indications of chilling appear on the surface of the molten bath, and see in general that the process of filling the molds proceeds in a proper manner. As soon as this operation is concluded, a few shovelfuls of sand should be thrown over the surface of the slabs to prevent sudden and unequal chilling. By the time the new charge is in the furnace and the assistant is at liberty to attend to his bricks, they will usually be found ready for removal, though still at a red heat on the surface and in most cases quite liquid in the interior. It is essential that they be removed, and the slight roughnesses that arise from the broken ends corresponding to the gutters through which they were filled be trimmed off with a small cutting hammer while they are still quite hot, as it is just at this stage that they possess the highest degree of toughness, and permit of manipulations that, if they were cool, would inevitably break them into fragments. These slabs are best removed from the furnace by being loaded upon the low iron barrow commonly used for the transportation of pigs of slag and matte. The loading is effected by means of a long five-eighths inch iron rod, bent into a hook at one end, and the blocks are then wheeled out upon the dump, where a special workman trims them properly, rejecting all that are imperfect or already cracked, and when cool, piles them into rows, to remain until needed. The most useful size for general purposes has been found to be about 8 by 10 by 20 inches, and weighing about 85 pounds; but by simply changing the form of the pattern, they may be produced of any desired shape or size, although experience has shown that it is not economy to attempt the manufacture of very thin slabs, or of any weight below 45 pounds. The immense value of this building material, produced from an otherwise worthless substance, and obtainable in rectangular shape for plain walls and foundations, in wedge shape for arches and for forming a circle in walling wells, and for many other daily needs, can be fully appreciated only by those who have had occasion to build in a country where rock was unobtainable and brick poor and expensive.

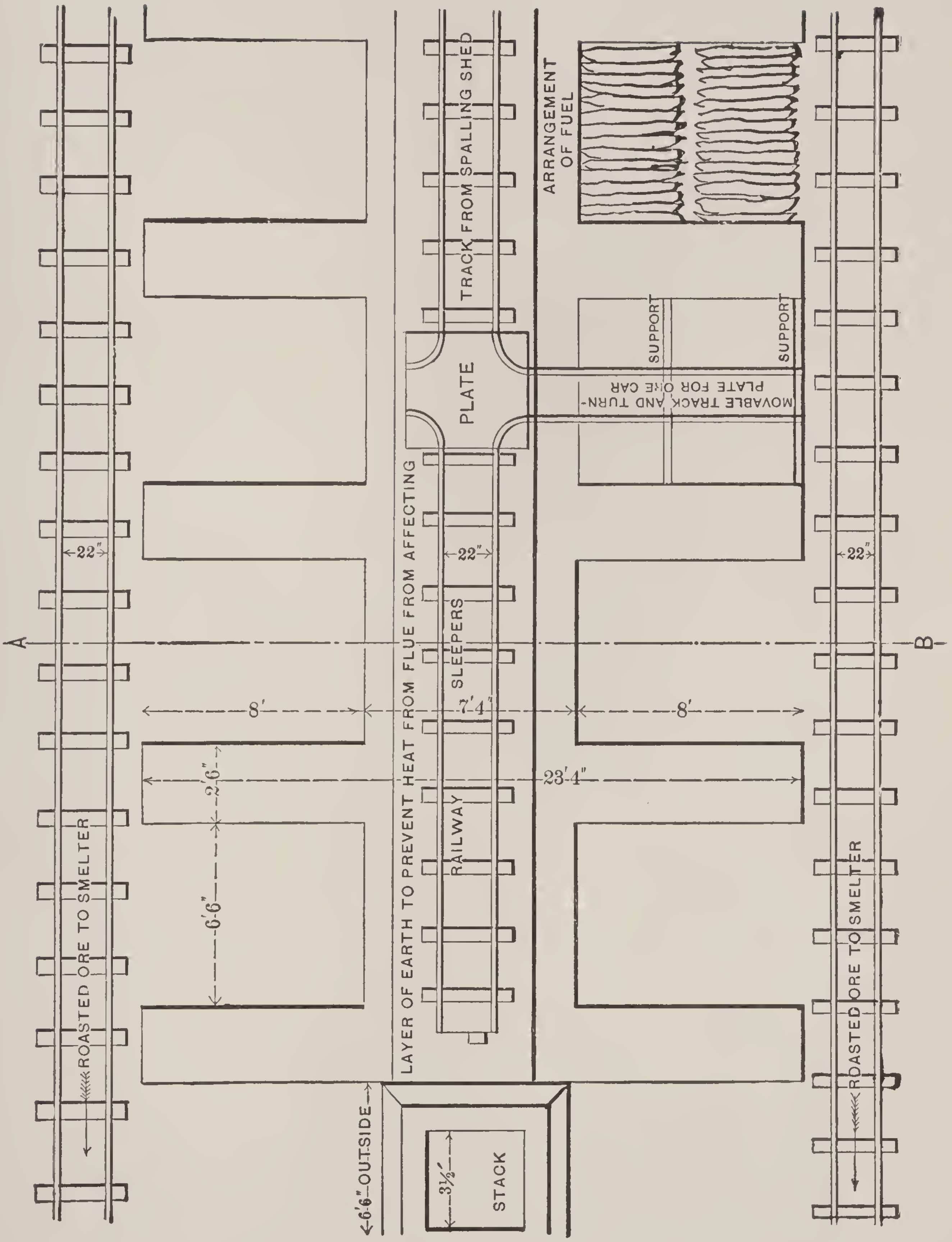
A particular distinction should be made between the old plan of making slabs of slag in iron molds, as practiced all over the world, and this method of sand molding, for which the profession is indebted to Mr. J. E. Gaylord, Secretary of the Parrot Silver and Copper Company. The author is well aware that molding in sand has been practiced also, but never, so far as he knows, with such results as are obtained by the method indicated.

To return to the roasting stalls. Assuming that they are to be built of the material just described, and without any iron-work for anchoring, and that each stall is to burn a charge of 20 tons and be again cleared out in 10 days, thus furnishing 2 tons a day, it will require some 56 stalls to furnish 100 tons of ore a day, allowing some 12 per cent. in excess of the needful capacity to permit of repairs. The weight of ore as brought to the stalls, and not as *taken from* them, is counted: its loss during the process of calcination depends upon the quality and amount of sulphides present, and frequently reaches 15 per cent., though a considerable portion of the loss in weight due to the elimination of the sulphur is offset by the gain in oxygen.

ROAST STALLS FOR ORE.


CHIMNEY
3'6"
ORE CAR
CROSS SECTION THROUGH A.B.
SCALE ⅛ IN. = 1 FOOT
a.a.—Draught holes connecting with Flue in sidewalls.
b.b.—Flue holes into Main Culvert.
a
b
24"
MAIN CULVERT
6'
TRACK TO SMELTER
TRACK TO SMELTER
8'
2'8"
2'
2'8"
8'
23'4"

Such a battery of stalls should always be built in a double row, back to back, each lateral wall serving as the division between



ROAST STALLS FOR ORE.
SCALE 3/8 IN. = 1 FOOT

the two adjacent partitions, while the unbroken rear walls form the sides of the main flue, a space of at least 2 feet being left between them, which simply requires a 4-inch brick arch to form the main flue for the entire system. This also constitutes a foundation on which, after a little leveling up with earth, to prevent the sleepers from being affected by the heated masonry below, the narrow railroad is laid on which the ore for roasting is brought to any part of a given stall by means of the turn-plate and movable rails, explained in the chapter on Heap-Roasting. A double row of 28 stalls (56 in all) should have a flue at least 2 by 4 feet for the third of the number nearest the chimney, which may be reduced to 2 by 3 feet for the middle, and 2 by 2½ feet for the end third, if any saving can be effected thereby. The two long rear walls, inclosing the main flue, should be 32 inches thick—once and a half the length of a slag brick—with proper allowance for mortar and irregularities, and should be laid solely in clay mortar, which designation throughout this entire work may be understood to mean merely common sticky mud, such as is employed for making a poor quality of red brick. If ordinary clay be accessible, it may be mixed with sand in such proportions as to slip easily from the trowel; otherwise, any ordinary sticky mud may be used, and will be found to form perfectly satisfactory material for laying all mason work that is to be exposed to sulphur fumes and a heat not exceeding a dull red.

The fact that lime mortar is totally unadapted to ordinary metallurgical uses, although doubtless universally known, is for some unaccountable reason frequently not acted upon, and the result in most cases is the rapid and total destruction of the furnace-arch, chimney, flue, or whatever structure may happen to have been put together with such unfit material. The acid vapors immediately form a sulphate with the lime present in the mortar, and this, absorbing water, becomes gypsum and crystallizes, expanding with great force, breaking the joints, and soon crumbles and washes away. It is quite proper to use lime mortar in such portions of the structure as are free from contact with heat and sulphurous gases, and yet require unusual strength, which cannot be obtained with the clay substitute. Such, for instance, as in the construction of chimneys for metallurgical purposes, where the best results can only be obtained by the employment of both of these substances: lime mortar for the outside work, while the common clay mud is merely used for the inside layer, and the joints thoroughly protected against any invasion of the sulphur gases by plastering the whole interior with a thin coating of clay mortar, tempered with sand to such an extent that it will not crack and fall off in sheets. Further reference will be made to this point in the chapter on Furnace Building. The constant and flagrant violation of this law is a sufficient reason for its frequent reiteration. A recent example suggests itself, where the arches of a number of very expensive and nearly new calcining-furnaces had fallen in, causing a very heavy loss. A conversation with the mason who built them brought out the fact that they were constructed with lime mortar, he having had no orders to the contrary.

The size of the stall is somewhat dependent upon the quality of the ore to be roasted, a highly siliceous ore with a comparatively low percentage of sulphur permitting a much wider and higher stall than an ore with little gangue, and especially than one containing a considerable portion of iron pyrites, in which case extensive and unavoidable sintering will follow any attempt at increasing the size of the stall. A safe size for an average ore, containing a moderate amount of pyrite and demanding careful handling, is 8 feet in length by 6 feet in height by 6½ feet in width, the latter measurement being the maximum that is safe under any ordinary circumstances. It is best to build the lateral walls of the same thickness as the rear division, the increased stability and durability of the entire structure well repaying the slight additional expense in labor and material. The bottom should be paved with the same slabs placed flatwise and exactly reversed from the position in which they lay when formed; their upper surface now going downward, while their original lower surface, which should be perfectly smooth and level, now comes upward. The connection with the main flue is effected by means of 8 or 10 small rectangular openings—2 by 6 inches—in the rear wall, in two or more rows, and at a considerable distance from the ground. These are kept tightly closed by means of a bunch of old rags or a ball of clay, when there is no occasion for their remaining open; otherwise, the draught of the entire system might suffer.

The only air admitted to these stalls originally, at the Parrot works, came through such interstices as were left in roughly building up the temporary front wall; but experiments led to the addition of some 4 or 6 similar openings in each lateral wall, which did not communicate with the main culvert, but connected with the outside air by means of a small flue running longitudinally through each division wall, though not extending so far as the central passage. This innovation has been followed by a decided improvement in the oxidation of the ore and a great diminution in the amount of matte produced. An essential precaution in the management of these stalls is to maintain a thick coat of clay plastering over their entire interior surface, by which the heated ore is kept from sticking to the walls and causing the rapid destruction of the mason-work. A few moments' attention to the empty structure after each operation will keep the plastering intact and greatly lessen the cost of repairs. As the entire success of this process depends upon the strength and regularity of the draught, a stack of considerable size and height is essential.

A battery of 56 stalls, as described, requires at sea-level a chimney 75 feet high, and with an internal area of at least 9 square feet, as will be further explained in the chapter on the construction of calcining-furnaces. Any economy in the direction of diminishing the size of this important adjunct will be immediately noticed in the lengthening of the roasting process, and may reduce the capacity of the stalls to an incredible degree. The draught is regulated by means of a sheet-iron damper hung in the main flue, close to its junction with the chimney, while the same office is accomplished for individual stalls by partially filling the draught-holes in the rear wall with bits of bricks or balls of clay. In no portion of the process, are the skill and care of the roasting foreman better displayed than in his management of the draught, which must be varied according to the season, and temperature of the air, as well as with the changing character of the ore.

As already intimated, a stall of the size and pattern just described will hold about twenty tons of pyritous ore, which should be kindled with the very smallest possible quantity of wood that will set it thoroughly on fire. This is essential for a far more important reason than the mere saving in fuel; for the slightest increase in the contents of the bed of wood on which the rock is heaped will, with pyritous or otherwise easily fusible ores, cause an amount of sintering and a formation of matte entirely disproportionate to the cause. Repeated trials can alone determine the various minutiæ of this description essential to the best possible results with the material under treatment; but, in most cases, where the ore is at all pyritous, good sound wood will be found to produce too fierce a heat for the purpose, and recourse must be had to decayed wood, which can usually be obtained at from one-half to two-thirds of the price of the sound fuel. For an ore containing 30 per cent. sulphur and, say, 25 per cent. silica, 25 cubic feet of rotten wood, or about one-fifth of a cord, will be found ample; but this small proportion of fuel—only one-hundredth of a cord to the ton—must be utilized in a proper manner, and with the most rigid economy and exactitude, or the heap will miss fire completely, doubling the cost of the operation, as well as interfering with the estimated production of the plant. A quarter of an hour spent in watching the manipulations of an experienced roaster is better than pages of description, though the operation of preparing a stall for its ore charge is far from complicated.

After seeing that the layer of clay on the inclosing walls is renewed with the plastering-trowel where necessary, and that the draught-holes are open to the extent dictated by former experience, a central longitudinal and two lateral flues are constructed on the floor of the stall out of large, irregular fragments of ore. These are merely to introduce air to the interior and to insure the rapid and thorough kindling of the entire structure. They are filled and surrounded with dry kindling-wood, and the greater part of the fuel, split into long, thin sticks from the large rotten logs and poles that are usually provided, is disposed in a thin layer over the bottom of the stall, the amount slightly increasing toward each side. The structure is now filled with coarse ore, and the ragging distributed throughout the entire contents rather than concentrated in a considerable layer merely upon the surface. As the stall becomes gradually filled, single small sticks of wood are placed between the ore and the lateral and back walls; while between the contents of the stall and the front wall, which is built up with large lumps of ore or stall matte, a considerable quantity of light wood is introduced to insure the thorough desulphurization of the anterior surface. A single car-load of ragging is spread on top of the coarse ore, and upon this a three-inch layer of shavings, bark, and chips is placed as a bed for about one and a half tons of raw fines, which, if disposed in the exact manner indicated, and covered thoroughly with well-roasted ore from a contiguous stall, will be thoroughly desulphurized, and the covering layer itself being in a well calcined condition, the entire contents, after burning, may be passed on to the next operation. Mr. R. Pierce, of Argo, uses with great advantage a sheet-iron cover over the top of his stalls, luted tightly with clay to the walls on which it rests.

It is only by employing great care, and after repeated trials, that the requisite skill will be attained to thoroughly calcine the large proportion of fines just indicated; but when one reflects that it amounts to some seven per cent. of the entire ore, and perhaps one-half of the total fines produced, it will be seen that the result is worthy of any pains that can be expended on it. The large pieces of raw ore that are employed in building the flues and front wall become gradually oxidized upon the surface, and slowly crumble away and mix with the finished product until they totally disappear and are replaced by fresh pieces. When the ore is to be removed, the front wall is taken down, and the lumps of ore from it are piled out of the way, to be again used for the same purpose.

The stall should be fired at night, as the smoke is so dense during the first few hours, and the draught so sluggish, that only a small part of the fumes find their way into the proper channel; but by the time the wood is consumed, the entire structure has become so much warmer as greatly to improve the draught. The sulphur and other products of volatilization and "sweating"—alluded to in describing the management of roast-heaps—form a sort of crust upon the surface, and seal all interstices connecting with the atmosphere, and force nearly all fumes to pass into the flue, thus greatly abating a nuisance. For the first twenty-four hours, the fire is confined to those portions of the ore that were in immediate contact with the fuel. The process of oxidation advances very rapidly, and by the close of the second day it is hardly possible to bear the hand upon the middle of the upper surface of the stall, showing that at least one-half the contents is already in combustion. By the end of the fourth day a similar degree of temperature may be felt upon the upper surface, at the very back of the stall, proving that the process has by that time invaded the entire length and breadth of the stall, though considerable time is still necessary for its thorough completion.

The successful progress of the process is clearly marked by the great rise in height of the entire contents, gaining some three inches in a single day, and frequently ascending some 12 inches above the level of the walls, at which it stood at the beginning of the operation, aside from the free space left to be filled out with ore from the disappearance of the fuel, amounting to some 25 cubic feet. This striking phenomenon, unfamiliar to those accustomed only to heap-roasting, where a settling rather than a rising of the entire mass occurs, is simply due to the fact that, in all cases of oxidizing roasting, a greater or less, though always very marked, increase in bulk occurs from the swelling and fissuring of the oxidized ore. The contents of the roast-heap, being perfectly free and unconfined, simply spread out laterally, while the consumption of the thick bed of fuel on which it rests detracts considerably from its height. The walls of the stall, however, inclose the ore in a rigid grasp, making it absolutely necessary that any increase in bulk, beyond that very slight amount necessary to replace the space occupied by the fuel, should take place vertically. In a badly burned stall, where extensive sintering has taken place, and a sufficient amount of the sulphides has melted into a solid mass to cause a decided diminution in bulk instead of an increase, the occurrence of crater-like depressions in the surface of the ore is positive evidence of such local fusions. That the pressure resulting from the increase in bulk is something quite tangible, may be inferred from the frequent pushing outward, or even overturning of the heavy lateral walls of a stall, provided one or the other of its contiguous compartments is either empty or unbraced, while the temporary front wall would inevitably be thrown down within the first day after kindling if not strongly supported by timbers.

The length of time necessary for the process under consideration is another uncertain factor. If the stall be left undisturbed, it will usually burn quietly for a period of twelve days, demanding little or no attention beyond an occasional shovelful of covering if heating too fiercely at any one point, and requiring about three days additional to cool sufficiently to remove with comfort; but, under ordinary every-day circumstances, no such moderation can be practiced, and the period of each operation can be curtailed, without any especial damage, to one-half this time. To accomplish this without detriment to the process of desulphurization, the following precautions must be adopted: As soon as the anterior surface of the ore is so cool as to impart no disagreeable sensation to the hand, the temporary front wall should be removed, the natural adhesion common to all sulphureted ores when roasted in lumps preventing the caving of the vertical ore face, which should be most carefully attacked with pick and shovel, every precaution being taken not to penetrate beyond the line of comparative cooling, and only so much ore being removed at any one operation as is consistent with the uninterrupted progress of the roasting in the mass behind. At least six or eight inches of ore should be left between the outer air and the line of fire, and any sudden elevation of the surface temperature, as well as increased difficulty in detaching the ore from the face on which work is prosecuted, is a sign to stop. To illustrate the ease with which the contents of a well-burned stall can be handled, the entire charge of ore from such a stall can be removed with nothing stronger than a shingle.

The first car-load is usually taken from the stall at the close of the fourth day, and the amount capable of removal may be rapidly increased, until in seven days more the compartment is again empty.

By this careful method of constant and systematic slicing, some two or three tons of well-burned ore may be taken daily from each of 40 or 50 stalls, and the capacity of the roasting plant rendered more than double what it would be if they were left untouched for the time necessary for their complete desulphurization and cooling; while the process of oxidation does not suffer in the slightest degree if the precautions just enumerated are adhered to.

In the case of ores containing arsenical pyrites, or, indeed, in the presence of any form of arsenical or antimonial combinations, a considerable proportion of the same that would otherwise go into the next operation in the shape of antimonates and arsenates may be volatilized and completely dispersed by the admixture of chips, small coal, brush-wood or other carbonaceous materials, which, as in heap-roasting, exercise a powerful reducing influence upon the products of oxidation just mentioned, and volatilize them in a metallic form. This simple precaution is of much greater value in the calcination of similar compounds in a pulverized condition in furnaces, where the different periods of oxidation and reduction are under the control of the operator, and can be made to follow each other in the manner most conducive to the object in view; but even in the rude process under consideration, experience has shown, in many cases, that a decided improvement in the grade of copper has resulted from this device, the simplicity and economy of which are among its strongest recommendations.

The results obtained in stall-roasting vary little as compared with those from burning in heaps. On the whole, it is not quite so easy to prevent the formation of matte in the former practice, nor do average and oft-repeated examinations show quite as good results in the elimination of the sulphur.

As circumstances may arise where it becomes the duty of the constructing metallurgist to decide between these two systems, to the positive exclusion of all methods involving the pulverization of the ore, and to give his reasons for and against each method, that his employers may also have some idea of the matter on which to base their advice or to rest the confirmation of his decision, it will be well concisely to review the comparative advantages and drawbacks of heap and stall-roasting.*

The first and most obvious advantage of the system of heap-roasting is the apparent cheapness and simplicity of the plant, only a level area being required, without furnaces, flues, stacks, or other expensive appurtenances.

The extreme simplicity of the method and the very satisfactory results obtained under proper management also speak in its favor; but further than this, no arguments can be advanced in support of the process.

Even the economy in first cost of plant will be found more apparent than real, when the expense of the trestle-work and track, as well as the establishment of the different grades between spalling-shed, roast-yard, and smelting-house levels are considered, and no one will deny the absolute necessity for such an arrangement if work on a large scale is to be prosecuted with any degree of economy.

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* See article on "The Mines and Smelting-Works of Butte City," by the author, in the United States publication on *Mineral Resources* (by A. Williams, Jr., 1885). The third method of roasting lump ore—that is, in continuous kilns—is only suited to certain peculiar conditions, and need not be considered when comparing the other two systems.

A careful comparative calculation of costs, corrected by the results of actual work, shows that, under ordinary circumstances, the difference in cost between the two plants under consideration is too trifling to have much weight in the choice of methods, and may even be on the side of the stalls in cases where the natural conformation of the land is unfavorable for the establishment of the terraces necessary for cheap heap-roasting.

A far more important reason for the adoption of the stall system is the great saving in time, by which the delay incidental to the cruder process of calcination is diminished by at least eighty per cent.

In works of large capacity, this becomes a question of vital importance; for few smelting companies are so amply provided with capital as to carry a constant stock of some ten thousand tons of ore, representing a money value of several hundred thousand dollars, which is not at all an extravagant estimate for works of the capacity under consideration. The circumstance that this amount may be reduced to a sum not exceeding one-fifth of the above by the substitution of the quicker method of calcination is a weighty argument for its adoption.

By a careful comparison of the expense of the two operations, we have already seen that a saving of about one-third may be effected by the use of stalls, owing principally to their greater economy in fuel and labor.

A still further advantage may be claimed for them in the concentration of all noxious fumes into a single flue, and their discharge into the atmosphere at such an elevation as to insure their gradual diffusion and dispersion without annoyance or damage. This is a great boon to the surrounding country, and more especially to the workmen employed in the process of roasting, as any one familiar with the atmosphere of an establishment where heap-roasting is practiced can testify.

Still further may be mentioned the considerable saving effected by the thorough roasting of the entire contents of the stall, including even the fine covering material, all of which is in condition for the succeeding operation; whereas, in the case of heap-roasting, at least 10 per cent. of the entire stock requires a second handling. Here may also be considered the serious losses of metal from wind, rain, and other atmospheric causes, which, although not entirely obviated by the employment of stalls, are at least greatly lessened; the saving in a certain plant of moderate capacity amounting in a single year, according to the author's calculations, to more than sufficient to cover the entire cost of erecting the stalls.

But the most important advantage possessed by stall-roasting over heap-roasting in an ordinarily moist climate—if the process be carried on in the open air—is the prevention of loss by leaching.

We have already pointed out the necessity of guarding against this loss by every possible means at our disposal; but even with every care a considerable loss from this source cannot be avoided in any ordinary climate.

Mr. Wendt* gives some important figures bearing on this point, relating to heap-roasting as formerly practiced at Ducktown, Tenn., where, however, the rain-fall is exceptionally great. We quote also his estimates of cost, which, taking into account the low cost of fuel and labor, correspond closely with our own.

"Ore-roasting, as thus carried out (in heaps), was a very economical process in point of labor and fuel. On an average, one cord of wood was consumed for 40 net tons of ore for each fire. The cost of labor in the first fire was 5 cents per 1,000 pounds for both Mary and East Tennessee ores; for the second fire, 7 cents and 6 cents respectively were paid; and for fine ores, the pay was 12 cents per M.

"The exact cost per net ton of ore was as follows:

$\frac{1}{20}$ cord wood, at \$3	\$0.15
Labor, 1st fire	.10
Labor, 2d fire	.14
Materials	.03
Total per ton	\$0.42

"The losses of copper in the above-described roasting have been very generally ignored in judging of its expense. At least, proper emphasis has never been laid on them.

"Owing to an unexplained difference of several hundred thousand pounds between the fine copper produced at the Ducktown smelter during a period extending over several years, and the monthly fine copper statements arrived at by deducting one and one-quarter unit from the assay value of the ores produced, the writer's attention was forcibly called to this subject. A careful series of experiments was instituted; the results were rather startling. Repeated analysis of ore weighed into a roast-pile, and analysis and weighing of this same ore when sent to the matte furnaces, proved an almost incredible loss.

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* See *The Pyrites Deposits of the Alleghanies*, by A. F. Wendt. New York, 1866, page 19.

"From the large number of experiments and analyses, I quote the following striking examples:

Pile No. 349.—Mary Ore.

Gross weight of ore.	Per cent. water.	Per cent. copper.	Fine copper, pounds.
399,213	2·5	5·0	19,461
204,444	2·0	5·8	11,620
95,182	3·8	5·0	3,617
8,663	3·0	5·1	428
34,165	6·0	4·0	1,284

741,667 pounds raw ore contained 36,410 pounds copper.

"The pile after roasting weighed 741,716 pounds—assayed 3·31 per cent. copper—equivalent to 24,985 pounds fine copper; 11,125 pounds copper, or 31·4 per cent. of the contents of the pile, had been lost while roasting; 170 days were consumed in roasting the ore, and 69 days in removing it to the smelting-furnaces. Hence, the ore lay exposed to the weather for 239 days, that is, eight months.

Pile No. 447.—Mary Ore.

Gross weight of ore.	Per cent. water.	Per cent. copper.	Fine copper, pounds.
172,882	3·0	4·7	7,881
1,532	5·5	6·3	91
198,800	2·0	4·5	8,767
32,178	4·0	5·3	1,637
26,865	5·5	4·6	1,167
32,245	3·0	6·2	1,939

464,505 gross pounds ore contained 21,482 pounds copper.

"Weight of the roasted ore was 495,566 pounds, assaying 2·85 per cent., or 14,152 pounds fine copper. During an exposure of 186 days, the ore had lost 34·3 per cent. of its copper.

"All the experiments made on a total of nearly 3,000 tons of ore proved, beyond possibility of doubt, an average loss of more than one unit of copper, or over 20 pounds of ingot per ton of ore. This great loss during the roasting readily accounted for the deficit in the copper production, if only 1¼ per cent. was deducted from the assay value of the ores for losses by treatment. The actual loss by the smelting process, as practiced at Ducktown, approached two units. Further experiments were made to confirm the results obtained. Experiments in roasting in furnaces proved that no copper escaped in the fumes. This, indeed, was anticipated, as the heat in roasting never could reach a point at which copper is volatile. The only other possible loss is by the leaching of the roast-piles during the heavy rains frequent in the Ducktown hills; and to this cause the great losses were finally ascribed. In referring to experiments in the leaching of these ores later on, this subject will be discussed in detail. Suffice it here to say, that with a roasting in one fire only, from 1 to 1½ units of copper became soluble in water. The results were further confirmed by copper found in large quantity in the clay 'bottoms' of the roast-piles. After a shower of rain, the roast-yard would be covered with pools of green water highly charged with copper."

The cost of heap-roasting was estimated at 80 cents a ton, including the transportation of the ore both to and from the roasting ground, as well as its weighing and other slight manipulations. The expense of roasting in stalls may be safely placed at 54 cents a ton, a figure based on the actual treatment of many thousand tons of ore by this method.

The cost of a battery of 56 stalls, built in the manner recommended, and reduced to the standard table of prices adhered to throughout this work, is appended. Their life, under ordinary treatment, will not exceed six years, at the expiration of which time they will be found in such a condition as to demand complete rebuilding, although, of course, the stack will outlast many generations of stalls.

ESTIMATED COST OF 56 ROASTING-STALLS, EXCLUSIVE OF STACK.

This being the first estimate yet given pertaining to the construction of any considerable portion of a smelting plant, the quickest and most convenient method of arriving at the desired result will be presented a little more in detail than may be considered necessary in subsequent calculations.

The total expense of the finished stalls may be conveniently divided into the following heads:

1. Excavation for foundations.
2. Cost of slag-brick, clay, and other building materials, delivered on the ground.
3. Labor in building the stalls.
4. Total expense of the railroads belonging to this part of the plant.
5. Miscellaneous expenses and superintendence.

The actual expense of building a plant of this description will almost invariably be found much greater than the most carefully prepared estimates would indicate, unless the figures were made by a man of long experience in these matters. The value of the numerous estimates of cost and expense contained in these pages is principally due to the fact that they are, almost without exception, taken from the results of actual work, executed under the superintendence of the author. They may, consequently, lay claim to a usefulness and reliability that the most carefully prepared estimates of cost would not possess unless derived from, or at least corrected by, a long and thorough personal experience in such matters.

To prepare the foundations for the required number of stalls, assuming the ground to be comparatively level, will require about 60 days' labor, aside from the removal of the earth. This allows for an 8-inch pavement, and for an extension of the foundation walls about two feet under ground.

1. *Excavation for foundations:*

Labor, 60 days, at $1.50	$90.00
Removing the excavated material	35.00
Superintendence and miscellaneous extras	32.00
Total	$157.00

In order to estimate the amount of building material required, it is essential to determine the cubic contents of all the walls inclosing the 56 stalls, 28 in each row. The stalls being 6½ feet wide, and all walls being 32 inches thick, it will be seen that the entire length of the two main rear walls is 520 feet, to which must be added the aggregated length of the 58 partition walls, each 8 feet long, = 464, or a grand total length of 984 feet. This wall being 6 feet high and 32 inches thick, contains in round numbers 15,700 cubic feet. To this must still be added about one-third, to allow for the foundation walls, and also the necessary amount of slabs for paving the stalls. The details are as follows:

Main walls	15,700	slag-brick.
Foundation walls	5,250	" "
Paving	2,080	" "
Total	23,030	" "

As these slabs are 8 by 10 by 20 inches, they contain very nearly a cubic foot each, and when the very coarse joints that they form are also considered, it will be found that their customary rating of a cubic foot each will be perfectly safe. They are laid entirely in ordinary clayey loam, which may be found almost everywhere, and which, if too sticky to leave the trowel, will be greatly improved by the addition of one-fourth or more of sand, or even sandy loam. At our standard of prices, $1 per ton will be ample for such material, and will lay one hundred brick. The cost of the slag-brick has been placed at two cents on the ground, as their delivery is at least as expensive as their manufacture. The sum mentioned, that is, two cents apiece delivered, or one cent at the furnace, will cover the cost of making and trimming, and leave enough margin to occasionally replace the pattern blocks and other material necessary for their production.

2. *Cost of materials for mason-work:*

23,000 slag-brick, at 2 cents	$460.00
235 tons clay, at $1	235.00
Mortar-boxes, hods, screens, etc.	45.00
Total	$740.00

The persons employed for this work should on no account be the regular, high-priced brick-masons, as these fare but badly in handling the heavy, brittle slabs, and neither like the work nor are able to earn the large wages that they invariably demand and receive. The proper mechanics for this work are what are popularly known as "country stone-masons," whose apprenticeship at building stone walls, underpinning barns and houses, etc., has exactly prepared them for handling such rough and heavy material as that under discussion.

Experience in this particular kind of construction has shown that the most advantageous distribution of the force is to provide each stone-mason with two immediate helpers, who assist him constantly, bringing the slab, placing it in position, and, in fact, doing everything excepting the spreading of the mortar and that last wedging and chinking that are of such vital importance in the proper execution of work of this description.

There are no hod-carriers, as the slabs are delivered by wagons at the point most convenient to the workmen, and the mortar, easily and rapidly manufactured from the materials already mentioned, is brought in large pails, being used in immense quantities in work of this description, although every crevice should be well filled with small fragments of rock or slag, called "spalls."

It has been found that each group of three men, as described above, will lay on an average 100 slag-brick daily, and not more.

3. *Labor in building stalls:*

Estimate for laying 100 brick:

One stone-mason	$3.00
Two laborers at $1.50	3.00
Mixing mortar for same	.50
Carrying mortar and other miscellaneous labor	.15
Superintendence	.35
Total for 100	$7.00
Total for 23,000 brick	$1,610.00

4. *Cost of railroad tracks.*—As all railroads about the works should be of the same gauge and pattern, a single detailed estimate will determine the cost per foot once for all. For tracks of the required description, having a 22-inch gauge, and calculated to carry a net load of 1,800 pounds, the car weighing an additional 800 pounds, a good quality T-rail of not less than 12 pounds to the yard should be selected and well fastened in place by a spike in every sleeper, while the abutting ends of the rails should be firmly secured by fish-plates, tapped for four $\frac{5}{8}$-inch bolts, two to each rail. Unless the bolt-holes in both fish-plates and rails can be bored where ordered in such a manner that there shall be no doubt of their perfect correspondence, it is

better to leave the plates blank, and bore them on the spot. This may seem a slight matter, but its neglect sometimes causes serious annoyance and delay in outlying districts, and the boring of the thin fish-plates is a slight task, as every smelter should be provided with a boring-machine run by power, which is indispensable for sampling pig-copper, and will be found generally useful.

The sleepers are sawed from the ordinary timber of the country, and may be conveniently ordered of the following dimensions: 36 inches long, 6 inches wide, and 4 inches thick—containing each 6 feet, board measure. They should be placed 39 inches apart from center to center, and last almost indefinitely, as the sulphate salts with which they become impregnated prevent their decay.

For convenience of calculation, the estimate will be based on a length of 100 yards of track:

Weight of iron:		
200 yards rails at 12 pounds =	2,400 pounds	
Spikes, bolts, and fish-plates =	115 "	
Total	2,515 pounds × 3½ cents =	$88.02
Sleepers:		
125 containing 6 feet each = 750 feet at $25 per M =		18.75
Labor:		
Grading, laying track, ballasting, etc.	$13.50	
Superintendence	6.00	
Total		19.50
Average allowance for curves and switches		13.63
10 per cent. for incidentals		14.00
		$153.90

We may therefore accept the figure of $1.53 per yard, or 51 cents per foot, as the cost of a tram-road of this description, and there being three lines of track required for the stalls, aggregating a length of 780 feet, to which must be added 100 feet for connections, switches, and single main line to smelter, we have a total of 880 feet at 51 cents = $448.80.

Rails for long curves may be bent cold; for short curves, they must be slightly heated; while frogs, points, etc., require welding, and can be readily constructed in any ordinary blacksmith's forge.

Great care should be taken in laying the track, nor should the foreman rest satisfied until every point, frog, and guard rail is in proper position and has the precise curve necessary for easy passage of the car without undue friction or danger of derailment. It is scarcely necessary to say that this work can only be properly and economically executed under the direction of an experienced railroad constructor.

5. *Miscellaneous expenses and superintendence.*—Aside from the allowance made in each department of the work for the above purposes, it will be found in practice that a considerable additional sum is required to cover errors in construction, blacksmith work, and various incidentals, as well as general superintendence, amounting in a case similar to the above to

	$211.00
Cost of 4-inch brick arch to cover main flue	137.00
	$348.00

Summary.

Excavation for foundations	$157.00
Materials for mason-work	740.00
Labor in building stalls	1,610.00
Railroads	448.80
Miscellaneous and superintendence	348.00
Grand total	$3,303.80

Uneven ground, bad weather, and other unfavorable causes may increase this sum to a considerable extent, but the figures given will be found safe under ordinary circumstances and with strictly judicious and economical management.

The calcination of matte in ore stalls of the pattern just described is by no means impossible, the principal difference between its treatment and that of ore being the increased quantity of fuel required—about three times as much. A considerable proportion of the matte will be fused during the operation, and another large fraction scarcely affected by the process; so that from three to four burnings are required to effect any reasonably perfect desulphurization.

This practice cannot be recommended, as much better results are obtained by providing the stalls with grate-bars, and preventing the radiation of heat from the surface by means of an arched brick roof, as described in the succeeding chapter.

THE STALL-ROASTING OF MATTE.

This is a method well known in the Eastern States, and practiced first in this country, so far as any record can be found, at the old Revere Copper-Works in Boston, and in more modern times at Copperas Hill in Vermont, and at the noted Vershire mine in the same State, where some sixty or seventy stalls are still in use. The partial suppression of the excessively disagreeable fumes generated in the heap-roasting of this substance; a gain of at least one-third in the time of treatment—no unimportant item in the handling of such valuable material; and a very great diminution in the losses caused by the elements, are the principal reasons for the selection of stalls in preference to heaps. On the other hand must be placed a heavy investment in buildings and in the stalls themselves, with their flues, stacks, etc. The mere grate-bars for a single matte stall cost in the neighborhood of $75, and the constant repairs that are peculiarly necessary in the case of mason-work saturated with the products of volatilization, and racked by the frequent and extensive fluctuations in temperature, due to the ever-recurring heating and cooling of the interior, render them a somewhat expensive portion of the plant, as will be seen in detail in its proper place.

MANAGEMENT OF MATTE STALLS.

The grate-bars being thoroughly cleansed and freed from all clinkers and *débris* of the preceding operation, and replaced in position, and the brick walls forming the sides and back of the stall receiving a fresh coat of plaster (clay) where necessary, a layer of fuel is placed upon the grate-bars, and the broken matte thrown upon this by means of a closely-tined fork, to separate the fine stuff, which is scattered over the top after the stall is filled with an average charge of from five to six tons.

The fuel employed is wood in 4 or 6-foot lengths, and split to a comparatively uniform size. From 10 to 20 cubic feet are used for each charge, metal of low grade rich in sulphur requiring less fuel than the higher varieties of matte. Experience has taught the great advantage obtained by the use of hard wood, and too much care cannot be bestowed upon the selection of the fuel, which should be of the best quality and thoroughly seasoned, as the result of the operation depends to a remarkable extent upon the quality of the fuel used.

Matte of any grade, from the lowest coarse metal to the highest quality of regule, may be treated in these stalls with almost equal results as regards desulphurization.

The stalls are always covered by rude sheds, to protect the brick-work from the weather, and should be paved with slag blocks, flat stone, or, much better, heavy iron plates, as the constant hammering that it must undergo during the spalling of the matte and the breaking of the huge clinkers that form an almost necessary accompaniment of this process, quickly destroys any other description of pavement.

The results of desulphurization by this method being no more thorough than by heap-roasting, the same number of burnings is necessary as in the latter case, and, owing to the difficulty of removing the heavy clinkers from the walls and grate-bars of these little furnaces, as well as the constant bill of expense for repairs, the cost of the process is about the same as in heap-roasting. The almost complete identity of the two methods in this respect renders any further details of expense unnecessary.

The imperfections of all the methods of roasting matte in lump form, as well as the great waste of time and metal, and the annoyance caused by the fumes, are serious objections, and it is only under exceptional circumstances that these crude and dilatory methods can be recommended. In nearly all advanced works, they have given place to the much more rapid and perfect method of calcination in reverberatory furnaces.

The ordinary dimensions of the stalls in use, now or formerly, at some of the principal works in this country are as follows:

Width	5 feet.
Depth (front to back)	6 feet.
Depth of ash-pit	1 foot 6 inches.
Height from grate to spring of arch	4 feet 8 inches.
Thickness of division walls	1 foot 4 inches.
Thickness of rear walls	1 foot 8 inches.
Area of flue opening in rear wall	160 square inches.

A stall of this size will contain from five to six tons of matte, and will burn for four days at the first firing, and for about three days at each subsequent operation.

Where three burnings take place, the capacity of each matte stall may be placed at one-half ton daily, and the amount of wood required for the three burnings will be one-twelfth of a cord per ton of ore.

From the measurements already given, aided by the estimates for brick-work found in a succeeding chapter, the cost of a block of such covered stalls may be easily arrived at; the covering arch consisting of a 9-inch semicircle of red bricks, and the main flue section being at least equal to the combined area of the flues that enter it.

The anchoring of a block of such stalls is very simple, consisting of longitudinal $\frac{3}{4}$-inch rods, while the uprights may be iron rails or stout wooden timbers. Each side wall should also be braced from front to back in the usual manner, while the front wall of the stall is a temporary structure of brick laid loosely upon the grate-bars and braced with a few lengths of flat iron. Fire-brick are ordinarily used for this purpose, the common red brick of which the entire permanent portion of the structure is built being too light and fragile to stand the repeated handlings and the fluctuations of temperature.

Since the ordinary charge only fills the stall about two-thirds full at the front, and slopes up against the rear wall to nearly the height of the flue opening near the top of the walls, or even in the arched roof, a large space exists between the upper edge of the temporary front retaining wall and the high semicircular brick roof. Through this, the sulphurous fumes and the products of the combustion of the fuel during an early stage of the process escape in such clouds as to render the atmosphere of the shed unfit for respiration. To partially obviate this difficulty, a sheet-iron curtain, suspended by wires running over a pulley in the roof, and furnished with a counter-weight, is used, and if properly fitted and luted to the side walls with a paste of stiff clay, is of great service.

It may be assumed with safety that, by the process of matte-roasting in lump form—whether executed in heaps or covered stalls—from two-thirds to three-fourths of its original sulphur contents is eliminated, by not less than three consecutive burnings.

CHAPTER VI.

THE ROASTING OF ORES IN LUMP FORM IN KILNS.

By the term kiln, as used here, we understand a comparatively small, shaft-like furnace, provided with a grate or opening for the admission of air from the bottom, and connected with a draught flue. The action is a continuous one, and the necessary heat is derived entirely from the oxidation of the sulphur and the other constituents of the ore.

No other class of furnaces has received greater attention or been brought to a greater state of perfection; but it is as an adjunct to the manufacture of sulphuric acid rather than to the calcination of ore that this apparatus must be esteemed, and consequently to the works treating on that subject that we must look for detailed descriptions and estimates of the same. The student is referred to Lunge's exhaustive work on Sulphuric Acid for full details of construction and management.

While the various processes of roasting hitherto described are suited to almost every variety of sulphureted copper ore, and yield equally good results whether the percentage of sulphur and copper is small or large, a much closer selection of material is indispensable for successful roasting in kilns, and their range of usefulness is restricted to comparatively narrow limits.

This very question of selection, however, varies greatly with the purpose in view, and depends upon whether it is desired merely to desulphurize a given ore without any attempt to utilize the volatile products of oxidation, or whether the manufacture of sulphuric acid is to be combined with the process of roasting.

The conditions necessarily present before any pyrites can be utilized for the manufacture of sulphuric acid are of two kinds, commercial and technical.

The commercial conditions are sufficiently obvious to any thoughtful mind, and are very plain, such as sufficient supply of ore at a fixed and low rate for a reasonable length of time, and contiguity to water, railroads, or some cheap means of transportation to the manufactory, which, owing to the nature of its product, must be situated in the immediate vicinity of its market.

The technical conditions, though more numerous, are almost equally easy of comprehension. An almost absolute freedom from gangue is essential, for the simple reason that the presence of foreign substances lowers the percentage of sulphur and necessitates the handling of worthless material, thus lessening the capacity of the works and producing other unfavorable results. For the same reason, though in a less degree, the presence of any other sulphides but the bisulphide of iron, which forms the ore proper, is disadvantageous; for no other compound of sulphur contains either so high a percentage of the same or parts with it so freely. Even the copper pyrites, which in many instances forms the principal value of the ore, is detrimental to the manufacture of sulphuric acid, both because it contains less sulphur and because it is too fusible to permit the proper regulation of the temperature. Beyond the limit of eight per cent. of copper in the pyrites, it cannot be profitably employed in the manufacture of acid. The Spanish pyrites, from which so large a proportion of the acid produced in England is made, contains on an average about three per cent. of copper, and about 48 per cent. of sulphur, this remarkably high percentage of sulphur showing its freedom from gangue.

An analysis of the average ore from the celebrated Rio Tinto mine may be of interest, as a type of a very favorable cupriferous pyrite for acid making:

ANALYSIS OF RIO TINTO PYRITES BY PATTINSON.

Sulphur	48·00	Magnesia	0·08
Iron	40·74	Arsenic	0·21
Copper	3·42	Insoluble	5·67
Lead	0·82	Oxygen and moisture	1·00
Lime	0·21		
Total			100·15

The ore used by three large acid-works in Boston and New York is obtained principally from Canada, some thirty miles from the Vermont line, and although somewhat variable in purity, averages about 3·5 per cent. of copper and 45 per cent. of sulphur, the percentage of gangue being greater than in the Spanish ores.

An excellent quality of pyrites is mined from a large deposit in Western Massachusetts, and in both Virginia and Georgia are beds of pyrites now under process of development, which, on competent authority, are said to rival the Spanish mines in almost every particular. The total product of pyrites for the United States in 1890 was 97,706 tons. There were also 115,000 tons imported from foreign countries, principally from Spain.

The presence of arsenic and antimony has a deleterious effect on the quality of the resulting acid, while lead heightens the fusibility of the charge, besides wasting sulphur by forming a stable lead sulphate, and any foreign substance, however harmless otherwise, lessens the percentage of sulphur.

An important point, sometimes overlooked by non-professionals in determining the value of a sample of pyrites, is its mechanical behavior during the process both of crushing and of roasting. A granular ore, soft or easily disintegrated, will increase the proportion of fines, which, although now utilized with great success in the manufacture of acid, are still undesirable as requiring a more expensive plant and entailing a greater cost in their treatment. A still more serious production of fines may take place in the kiln itself in the case of ores that decrepitate, sometimes occurring to such an extent as entirely to choke the draught and render their employment impossible.

One of the most serious errors ever perpetrated in the manufacture of acid from pyrites is the attempted employment of pyrrhotite or monosulphide of iron for pyrite—bisulphide of iron. Aside from the greatly lessened proportion of sulphur, 36 per cent. as against 53 per cent., the monosulphide will not even yield freely what sulphur it contains, but crusts with oxide of iron, turns black, and is soon extinguished when treated in an ordinary pyrites kiln. It seems scarcely possible that extensive works for the manufacture of sulphuric acid (and copper) should have been erected, their ore supply being entirely derived from a deposit of the valueless monosulphide; but such has been the case in more than one instance, and will continue to be so in enterprises conducted without the aid of skilled direction. One of the most striking instances of this kind is a now extinct Massachusetts company, which is said to have expended over $200,000 in this manner, all of which was a total loss, excepting the small amount realized from the sale of buildings and lands.

Under certain conditions, the use of kilns for the calcination of cupriferous pyrites without the production of sulphuric acid may be found advantageous, as in the case of the former Orford Nickel and Copper Company, near Sherbrooke, Province of Quebec, which, after employing heap-roasting for some time, erected a large number of kilns solely for the purpose of calcining its ore previous to smelting; finding the saving in time and avoidance of waste, combined with the lessening of the annoyance formerly experienced from sulphur fumes, a sufficient advantage to repay the somewhat heavy cost of the burners.

The minimum percentage of sulphur sufficient to maintain combustion in kilns does not yet seem to have been positively determined; but with an ore otherwise favorable, it is probable that 25 per cent. is quite sufficient for the purpose.

For economy's sake, as well as for the purpose of retaining the heat, kilns are constructed in blocks of considerable length, and of the depth of two burners, the front of each facing outward, while the flue in which the gas is conveyed to its destination is built on top of the longitudinal center wall. Fire-bricks are used wherever the masonry is exposed to heat or wear, and the entire block of furnaces is surrounded by cast-iron plates, firmly bolted in position, and provided with the necessary openings for manipulation.

No fuel is required after the burners are once in operation; and when in normal condition, the attendance demanded, aside from the labor connected with the regular charge of from 500 to 2,000 pounds of ore once in twelve or twenty-four hours, is very slight.

Much skill and experience, however, are required to maintain the regular working of the kilns, especially with ores that are not exactly suited to the process.

From five to ten per cent. of fines may also be desulphurized with the coarse ore without seriously interfering with the process. They are thrown toward the back and sides of the shaft, leaving the center uncovered; otherwise, the draught is affected and serious irregularities supervene.

In accordance with the policy adopted throughout this work, no detailed estimate of expense will be given in the few instances where the author is unable to base the same on personal experience.

Such is the case in kiln-roasting; but we are assured by the best authorities that the expense of calcination by this method does not exceed that of stall-roasting, though the first cost of the plant is considerably greater.

The results obtained by this process are unexampled in the roasting of lump ores, although there is no doubt that a considerable share of the success is due to the fact that the sulphur is the object of interest, instead of merely being a waste product to be driven off as far as convenient.

If more than 4 per cent. of sulphur remains in the *cinders*, as the residue from this process is called, the result is not considered satisfactory. It is needless to say that such a perfect desulphurization cannot be obtained in either heap or stall-roasting, nor is it necessary or, in many cases, even beneficial for the subsequent process, although, of course, in most instances the lack of sulphur in the furnace charge forms a welcome outlet for the admixture of raw fines, which may thus escape the expense of calcination.

Within the past few years, the utilization of these fines has attracted much attention, and the efforts to calcine them in automatic furnaces for the production of sulphurous acid have been crowned with success, as will be again alluded to when treating of the Roasting of Pulverized Materials.

The attempt to utilize kilns, with certain slight modifications, for the roasting of copper matte has, after many difficulties and much expense, attained a successful issue at certain European works, especially at the Mansfeld copper-works in Germany, the object in view being rather the abolition of the nuisance arising from the escape of the sulphur fumes into the atmosphere than any expectation of financial advantage from the employment of a substance so poor in sulphur for the manufacture of acid. It is obvious that only mattes comparatively free from lead and other fusible metals can be treated in this manner, and that the process of roasting is beset with difficulties that have only been overcome by the exercise of the greatest skill and patience.

CHAPTER VII.

CALCINATION OF ORE AND MATTE IN FINELY DIVIDED CONDITION.

Perhaps the most marked point of difference between the roasting of lumps and fines is the time requisite for their oxidation. Oxidation is almost instantaneous for an infinitely small particle of any sulphide, and the time increases with the cubic contents of the fragment, until such a size is reached that the air fails to penetrate the thick crust of oxides formed upon the outside of the lump of ore or matte, and all action ceases.

It might seem, therefore, that the process of pulverization should be pushed to extreme limits, and that the best results would be obtained from the most finely ground ore. But this is by no means the case in actual practice; for other conditions arise that more than counteract any advantage in time. The chief of these, aside from the difficulty and expense involved in the production of such fine pulp, are the losses in metal, both mechanical and chemical, that occur with every movement of the ore, and reach an enormous aggregate before the operation is completed; and the liability to *fritting* or sticking together in the calcining-furnace, regardless of the greatest possible care in this process. The oxidation of the particles takes place with such rapidity that a temperature is generated above the fusion-point of ordinary sulphides.

Still further objections could be mentioned; but those already adduced are sufficient to limit the degree of pulverization for the principal portion of the ore, although a greater or less proportion, according to the machinery used for the purpose, is crushed to an impalpable dust, and causes a considerable mechanical loss, in spite of all provision for its prevention.

The best size to which to crush varies with each individual ore, and is entirely a matter of trial and experience; nor should any one responsible for the calcination of any given material rest satisfied until he has determined by actual and long-continued experiment that the substitution of either a coarser or a finer screen for the size in use will be followed by less favorable results.

This may be arrived at by careful comparative determinations of the residual sulphur contents after the calcination of material crushed through screens of various sized mesh and roasted for the same length of time, careful consideration also being given to the cost of crushing in each case, to the condition of the oxides of iron present (the sesquioxide is an unfavorable constituent in reverberatory smelting), and, above all, to the quantity of flue-dust formed, and loss of metal by volatilization.

It is evident that such diverse and obscure questions can only be accurately determined by extensive and long-continued trials. But the result is well worth the labor, and in these days of almost universal information and close competition, it is only by such means that any decided advantage can be obtained.

While mattes, speiss, or similar products of fusion must always be granulated or pulverized to the degree required for calcination, it is not an uncommon quality of sulphide ores either to decrepitate, or else to fall to pieces when heated, by the mere moving from place to place in the furnace, to such an extent that the charge may be made up of pieces from the size of a walnut down, without affecting either the time requisite for the oxidation or for its perfection. The product will be an almost homogeneous and impalpable powder.

A more striking illustration of such a condition of affairs can hardly be found than in the case of the concentrates from the Parrot Company's mine at Butte, Montana.

In this instance, the process of subdivision resulted from two different causes. The iron pyrites that forms the larger portion of the ore decrepitates into very minute cubes, which are subsequently reduced to a fine powder by oxidation, while the fragments of pure copper ore—bornite—seem gradually to diminish in size by the wearing away of the surface as it becomes earthy and friable from the superficial formation of oxides.

This latter phenomenon may also be observed to a less extent in the calcination of mattes when they are of a sufficiently soft or porous nature; but in roasting a considerable quantity of a very low-grade matte (from 10 to 15 per cent. of copper) that had been obtained in hard polished granules by tapping into water, it was found impossible materially to alter either the size or shape of the grains, many of which were as large as an army bean, or satisfactorily to reduce the percentage of sulphur, even by long exposure to a temperature closely approaching its fusion-point.

On the other hand, quite satisfactory results are obtained in the case of richer matte (from 30 to 40 per cent. of copper) by granulation in water; and, in many of the foreign works, this is the only means provided for the preparation of the matte for the process of roasting; but it must be remembered that this practice is confined to the English reverberatory method, where it is not desired to remove more than 50 per cent. of the sulphur by roasting, and where a portion of sulphides still remains in the calcined matte that would be entirely unsuited to the so-called "blast-furnace" method of matte concentration in cupolas, as usually practiced in this country.

Although the results described, as obtained by granulation, may be improved upon by careful attention to the temperature and pitch of the matte when tapped, and especially by care and experience on the part of the smelter, this practice cannot be recommended, excepting under peculiar conditions and in remote situations, where improved crushing machinery is not obtainable, or where the physical condition of the matte is particularly favorable to the production of porous and friable granules. Nor is anything gained by its employment for the purpose of avoiding the preparatory breaker, and obtaining at once a material sufficiently subdivided for immediate treatment in the final pulverizing apparatus; for, although, in this practice, the larger granules are broken and crushed into a condition favorable for the calcining process, a large proportion of the entire mass is already so minute as to pass through the crushing apparatus untouched in the shape of minute spherical pellets or globules, which present the least possible surface to oxidation, and retain a hard, glossy surface. These grains are scarcely affected by any moderate temperature, and may even undergo complete fusion without any perceptible loss of sulphur. Not many years ago, the question of economy might have influenced the adoption of this practice; but at the present time, and in view of the improved and comparatively inexpensive machinery at our disposal, it is probable that the inconvenience, danger, and other drawbacks inseparable from the projection of large quantities of molten sulphides into water, and their subsequent recovery from the reservoir or whatever vessel is employed for the purpose, more than outweigh the cost of crushing by machinery.

It is impossible to lay down fixed rules for the degree of pulverization of any material best suited to roasting. Each case must be decided according to its own peculiar conditions, including the cost of labor and power, and the capacity and quality of the mechanical means available.

Bearing in mind the results that may, in certain exceptional cases, come from decrepitation, it may be assumed that a reduction in size beyond one-twelfth of an inch is seldom advantageous in treating ores, and that the presence of a large proportion of sulphides or of a particularly porous or friable gangue may permit an increase of the screen mesh to one-eighth inch or more. With mattes, a slightly finer standard (from one-twelfth to one-sixteenth inch) may be employed.

The proportion of the ore reduced to a minuteness neither intended nor desired depends materially upon the means employed for crushing; and as the mechanical loss and other evils enumerated increase in direct ratio to the amount of fine dust in the charge, it is evident that, other things being equal, the apparatus best adapted to the breaking of ore or matte is that which produces the smallest proportion of fines.

CRUSHING MACHINERY.

The crushing machinery used for the purpose under discussion may be divided into two classes

1. For preparatory crushing: Jaw-breakers of various patterns.
2. For final pulverization: Stamps, Ball pulverizers, Chili mills, various patent pulverizers and grinders, Cornish rolls.

I. MACHINES FOR PREPARATORY CRUSHING.

The jaw-crushers in almost universal use are eminently satisfactory as regards economy, capacity, and general suitability to the purpose for which they are intended. A few deductions from long-continued trials of almost every well-known pattern of breaker may be useful. Ordinary prudence will suggest to the inexperienced the choice of some form of machine long and favorably known to the public, and nothing can be more foolish than the selection of some novel and much vaunted but untried apparatus.

A machine should be selected that has stood the test of years, and is manufactured by some well-known and reputable firm. Light-built machines should be particularly avoided, as the strain exerted upon certain parts of every breaker, especially when clogged with clayey ore and set to crush fine without shortening the stroke of the jaw, is something enormous, and only to be successfully encountered by superabundant strength in every portion of the apparatus. This is well exemplified in the breakers turned out from the foundries of those manufacturers who have long made a study of this particular business, and who have gradually added an inch of metal here and a half inch there, as time and trials have developed the weak points of the machine, until it may appear bulky and clumsy beside the light and elegant models of some of their later competitors. Unless ore is delivered to the smelter in unusually large lumps, the 7 by 10-inch jaw-breaker will be found most convenient for general work, and not so heavy as to demand special arrangements for its setting-up. (These figures refer to the size of the opening between jaws—the smaller number indicating the distance between the fixed and movable jaw, while the larger gives the measurement at right angles to this.) Such a machine can be set to break to a maximum diameter of three-quarters of an inch, and has a capacity equal to any ordinary demands, although varying greatly with the size of the discharge opening and quality of the material crushed. The setting-up and management of this machine are matters of too universal knowledge to require further attention; but it may not be generally understood that the substitution of smooth jaw-plates for the corrugated ones usually employed will greatly increase the proportion of fines in the product.

As the ore usually passes directly from the breaker to the rolls—better with the interpolation of a short screen to remove such as is already sufficiently fine; and as in fine crushing the capacity of the breaker, even when set up to its closest practicable limits, usually greatly exceeds that of the rolls, a decided increase in the work performed can be most economically and easily effected by introducing a second fine breaker between the coarse crusher and the final pulverizer. This machine may be of quite light construction, should have a very long, narrow jaw opening—say 2 by 12 inches—a slight "throw," and move at a high speed.

It is in this direction that the most important improvements in fine crushing machinery may be looked for, and it is probable that the crushing and, still more important, the discharge area may be most advantageously increased by the employment of a multiple-jawed machine. This apparatus—when used merely as an intermediate crusher—will reduce the product of the coarse breaker to the size of corn, or even smaller, thus greatly lightening the work of the finishing pulverizer.

II. MACHINES FOR FINAL PULVERIZATION.

The apparatus at all suited for this purpose may be brought under the following heads:

Stamps.
Ball pulverizers.
Chilian mills.
Miscellaneous patent pulverizers.
Multiple jaw-crushers.
Cornish rolls.

Stamps, although universally known and always reliable, produce far too great a proportion of fine dust, besides being unnecessarily expensive, both as regards first cost and subsequent running.

The Ball pulverizer, when properly constructed, has the merit of compactness, slight cost, economy in running, and several other advantages, but is of insufficient capacity, and, like stamps, is better calculated for the production of fine pulp than of the material required for calcination.

Chilian mills have obtained a strong foothold in England for certain metallurgical purposes, but are expensive and cumbersome, have a very small capacity, and are peculiarly adapted to the production of impalpable dust.

The miscellaneous patent pulverizers now offered for sale would require a considerable space for their enumeration. In many cases, they possess much merit; but although differing to an extreme in almost every other particular, are pretty well united in producing a pulp containing too much dust in proportion to the granules.

Multiple jaw-crushers have already been referred to as promising much for the future. This construction admits of an enormous area of discharge opening, and since the breaking of each fragment of rock is accomplished by the approach of two opposing surfaces, which yet can never meet, all particles sufficiently fine are at once removed. There is reason to hope and expect that these machines will soon be perfected; in which case, nothing yet invented could be better suited to the production of just such material as is demanded both for concentration and for the variety of calcination now under discussion.

Cornish rolls.—No other class of machines can compare with the Cornish roll for capacity, economy, and certainty in crushing every variety of ore and matte for the purpose just indicated. But inasmuch as the various patterns of this machine differ almost as much among themselves in efficiency and capacity as they do from the other pulverizers already mentioned, and as an examination of a large proportion of the roller plants in actual use at the present time in this country indicates a great want of care in both construction and management, and tendency to be satisfied with a considerably lower standard of excellence than might easily be attained, it seems desirable to draw attention to such points as seem to particularly demand supervision or reformation.

Rolls should be ordered only from the best makers, who can refer to numerous similar machines of their manufacture in long and successful operation, nor should the metallurgical engineer forget that much of the work for which rolls are made, and in the performance of which they give perfectly satisfactory results, is for phosphates, gypsum, lead ore, or similar soft or brittle substances, whose crushing bears no relation to that of the low-grade matte and tough quartzose—or hard pyritic—ores that are generally the object of calcination. Certain low-grades of matte, especially when produced in blast-furnaces, contain a large proportion of various indefinite compounds of copper, iron, and sulphur that are almost malleable, and would inevitably destroy any of the ordinary light-weight, low-priced rolls so frequently considered sufficient for general purposes, and occasionally placed in metallurgical establishments with mistaken notions of economy.

The most important proportions, to be noticed in the type of rolls required for the purpose under discussion, are great diameter of the body of the roll in proportion to its face—2½ or, better, 3 to 1; great strength of axle, which may be with advantage one-quarter of the total diameter of the roll, including shell; great length and rigidity of bearings, which may be of Babbitt-metal, or, still better, of brass; proper size and weight of fly-wheels, and a general strengthening and reinforcement of all parts of the machine that are found weak or doubtful on comparison with the increased capacity of the portions just named. The frame in particular will require a decided augmentation of strength to correspond with the additions enumerated, and may advantageously be cast in separate halves, to avoid the inconveniences arising from its bulk and weight. The best material for the roll-shell, as well as the most convenient means for holding the two massive rolls in apposition, while provision is yet made for the passage of any infrangible or incompressible substance, are questions that have drawn out a variety of opinion and practice.

Two varieties of shell only demand consideration for the crushing of hard ores and mattes.

1st. White iron, chilled to the depth of nearly an inch, and so evenly that no variation in wear is detected on the surface after the passage of a thousand tons or more of the hardest material.

2d. Soft steel, such as is produced by the Siemens-Martin method at a very moderate cost.

Such chills as are here indicated, and as are alone satisfactory in practical work, can only be obtained by careful manufacture, and the ordinary chilled shells advertised as perfectly satisfactory by the greater number of manufacturers are comparatively worthless, and a source of constant annoyance and expense. A pair of chilled shells of 36-inch diameter, 14-inch face, crushed approximately 22,000 tons of medium ore, taking it from the fine breaker, 1½ inches, and reducing it to about five-eighths of an inch in size before being worn out, the chilled surface wearing smoothly and regularly away to the depth of over half an inch before any notable irregularities appeared.

It is always advantageous to have all the rolls in use in any establishment of the same diameter and make, as in this way a very great saving is effected by using all new shells for fine crushing, and when too much worn to yield economical results in this situation, to pass them on to the coarse crushing rolls, where they may perhaps serve an equally long time before being finally discarded.

It is by no means generally known that the hardest chilled iron may be turned with an ordinary tool without difficulty if a sufficiently slow motion is made use of in the process.* In this way, a set of shells may be preserved in condition for fine crushing for a much longer period than is usual, as the shells are almost invariably incapacitated for this purpose by their tendency to wear hollow in the center. This fault may be partially obviated by a simple device by which the stream of ore as fed to the rolls is diverted from the center and directed to the lateral portions of the roller surfaces; but is much more immediately and effectually remedied by turning the surface smooth under the precautions just indicated.

Steel shells also give excellent satisfaction, and are quite easily turned, as only soft steel is used in their manufacture.

No one who has watched the constant jumping of all ordinary rolls, with the accompanying separation of the opposing surfaces, and has noted the inevitable escape of a pound or more of coarse material through the crevice thus formed, can doubt the increase in capacity that would arise from the rigid fixation of the crushing surfaces at such a distance from each other that nothing could pass between them without being reduced to the desired size. Any proposition to this effect is usually met with the objection that the rolls would frequently become "stalled" for want of power, or else that constant breakages would arise from the passage of a bit of steel or some similar infrangible substance. The first objection is too trivial to require notice, while the second may be met in various ways: By either increasing the strength of the springs by which the rolls are maintained in apposition, to such an extent that no substance capable of being crushed without detriment to the machine can pass them without being crushed; or by abolishing springs entirely, and providing some weak point in the apparatus that shall break before the strain becomes dangerous to other and more expensive parts.

Both of these methods are in satisfactory use, the latter, however, being suited only to fine crushing; the employment of a thin cast-iron breaking-cup renders its application simple and economical; while the former improvement is best effected by the use of strong duplex steel car-springs, or heavy rubber springs, which are in use at many places, giving almost universal satisfaction.

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* The author desires to acknowledge his indebtedness to Mr. Franklin Farrel, of Ansonia, Conn., for this important point in the manipulation of chilled iron, as well as for many novel and useful suggestions in connection with rolls and crushing machines.

In erecting a new roller plant, provision should be made for convenience in changing the shells, which is a heavy and tedious task unless ample space is reserved overhead for the employment of block and tackle. A gain of several hours will be effected by having in readiness a duplicate set of roller-shafts as well as shells, so that each old roll may be lifted entire from its bearings, and its new substitute lowered at once into the vacant place.

Rolls frequently fail to meet expectations from being run at too low a speed. Seventy-five or even one hundred revolutions a minute for a 36-inch roll of the most modern construction is not too great, and can be used with no untoward results beyond the increased production of fine dust from the violent impact of two solid bodies moving at such a high velocity. Forty to fifty revolutions is enough for geared rolls.

Unless rolls are specially constructed for the purpose, nothing is gained in setting them so that their surfaces are in direct contact, even for the finest crushing, as they will constantly choke and give trouble, without yielding nearly as large an amount of product of the desired fineness as when they are set slightly apart, and the product that is not fine enough to pass the screen is returned to them.

Nearly all of the difficulties and annoyances experienced with elevators may be avoided by constructing them with a capacity greatly beyond their apparent requirements.

Strong, large cups should be selected, never less than ten inches in width, except for handling very fine material, and traveling at a rate of at least 280 feet a minute. These should be strongly riveted to a four-ply rubber belt, except in cases of perfect freedom from moisture, where leather is preferable.

Double chain elevators running over sprocket-wheels also do excellent work when properly made.

The feeding and management of the crushing plant should be intrusted to a careful and experienced man, any infraction of this rule being almost certainly followed by annoyance and loss.

Its capacity varies greatly with the quality of the material and the fineness to which it is crushed, diminishing very rapidly with the degree of comminution.

As a rough indication of what may be expected from the variety of plant just indicated, consisting of a 7 by 10-inch breaker, and a single pair of 36-inch rolls, with screen and elevator, the following figures are given from the author's note-book: A hard but brittle siliceous ore, carrying a small percentage of pyrites, was crushed through an 8-mesh screen at the rate of 2,236 pounds an hour. The substitution of a 12-mesh screen reduced the hourly production to 1,560 pounds.

A hard and tough matte was crushed through the latter screen at the rate of only 960 pounds an hour, nor can a much greater duty than the above be expected from any similar plant.

Any estimate of machinery and of such portions of the general plant as can be purchased ready for use, and consequently possess a specific market value, public to all, does not come within the scope of this treatise. Rolls are now made without gears, each roll being run by an independent belt. They give great satisfaction.

CALCINING FURNACES.

The furnaces suited especially to the oxidizing-roasting of sulphureted ores and mattes in a pulverized condition may be included under the following heads:

I. Shaft-furnaces.
II. Revolving cylinders.
III. Automatic reverberatory furnaces.
  a. With stationary hearth.
  b. With revolving hearth.
IV. Ordinary reverberatory furnaces.
  a. Open-hearth furnaces.
  b. Muffle furnaces.

I. SHAFT-FURNACES.

This group includes some of the most important and useful appliances for the roasting of sulphureted substances, where the utilization of the fumes for the manufacture of sulphuric acid forms a part of the process of calcination.

If the question of acid manufacture be left entirely out of consideration, and the comparative economy of each method of calcination be judged solely upon its own merits, it is doubtful whether resort would be had to these furnaces, save under exceptional conditions, as their limited capacity, great cost of construction, and imperfect work, except under the most skillful management, would effectually bar their introduction. But under the stimulus arising from the enforced manufacture of acid from pulverized pyrites, and the consequent necessity of employing some form of automatic furnace in which the gases arising from the oxidation of the ore are kept separate from the products of combustion of the fuel, this type of calciner has received such attention and study that it promises fairly to rival the most economical form of roasting apparatus known to metallurgy. The student is referred to Lunge's work on the manufacture of sulphuric acid for full details regarding this and other forms of furnace suited to the calcination of ores in connection with acid-making.

The Gerstenhofer shelf-furnace was the first successful calciner* of this type, and is still largely used, though becoming gradually supplanted by improved modifications. The few furnaces of this pattern that have been constructed in the United States have failed to answer the desired purpose, owing to imperfect construction, poor refractory materials, and want of skill in management. The Gerstenhofer furnace consists of a vertical shaft, surmounted by a mechanical device for feeding the pulverized sulphides in any desired quantity, and containing a great number of parallel clay ledges of a triangular form, one of the flat surfaces being placed uppermost. These are so arranged as to obstruct the ore in its passage, and delay it sufficiently to effect a certain degree of oxidation, which is seldom perfect enough to yield the desired result without a supplementary calcination in some other form of furnace. The front wall of the shaft is pierced by parallel rows of rectangular openings, for the purpose of changing the clay shelves or of cleansing the same.

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*As this treatise is intended to deal exclusively with American methods and practices, any detailed description of various valuable automatic calciners that in other countries have proved highly successful is omitted. Nor will consistency permit the elaborate description of certain American inventions—such as the Stetefeldt and Howell roasting furnaces—which, although invaluable for the chloridizing-roasting of silver ores, or even for the thorough calcination of fine pyrites for chlorination, have not yet been adopted to any considerable extent by the copper smelter. This is of the less consequence, as full descriptions of all these various forms of apparatus are accessible to the public.

The oxidation of the sulphides generates sufficient heat for the proper working of the process, so that the sulphurous gases may be obtained for the manufacture of acid free from any products of the combustion of fuel.

The Stetefeldt furnace, so invaluable for the chloridizing-roasting of silver ores, is a shaft provided with a grate for the generation of such a degree of temperature as would be lacking in the roasting of ores so poor in sulphur as those usually exposed to this treatment, as well as an auxiliary fire-place for the more perfect chloridizing of the flue-dust, which, owing to the fine pulveriztaion of the ore and the strong draught essential to the proper working of the apparatus, is formed in unprecedented amounts, and pretty thoroughly regained in ample dust-chambers.

The employment of an auxiliary fire-place, and the invention of a highly ingenious and perfect automatic ore-feeder, constitute important claims to originality that are frequently overlooked by writers in commenting on this furnace. Its capacity is very great, 60 tons in twenty-four hours being easily worked in one of the large-sized furnaces of this type; and were it possible to obtain equally good results by employing it for oxidizing-roasting, it would be the most valuable addition to the modern metallurgy of copper. But as it is at the present time, it cannot be enumerated among the resources of the copper smelter, although late experiments indicate the probability of its successful adaptation to this purpose.

The English acid-makers have introduced various modifications of the two last-named furnaces for the desulphurization of cupriferous iron pyrites. These may be found in Lunge's work, and are said to possess considerable capacity and yield excellent results.

II. REVOLVING CYLINDERS.

These also are extensively and advantageously used for the chloridizing of silver ores, having a considerable capacity, and effecting a thorough chloridization at a very moderate cost. They consist essentially of a horizontal or inclined brick-lined iron cylinder, revolved slowly by gearing, and having a fire-place at one end—or at both ends, used alternately.

Numerous experiments as to their applicability to the oxidizing-roasting of pyritous ores have been carefully carried out, and, while the author has not found it economical for this purpose in its original form, late experiments have shown its perfect adaptation to this class of work.

A still further advance has been made by Mr. James Douglas, who, by introducing an interior central flue for the passage of flame and smoke, and carefully graduating the supply of air, has combined the advantages of the revolving cylinder and the muffle furnace.

Since publishing the first edition of this book, great advances have been made in calcining ores in powder form, in automatic furnaces.

Both the O'Hara plow furnace and the shelf furnaces for acid-making have their advocates; but there is little doubt that, at the present time, the modified and improved Brückner's cylinder stands pre-eminent as the most satisfactory and economical of all automatic calcining furnaces for pulverized ore.

In describing this furnace, it would be unjust not to mention Messrs. Fraser and Chalmers, of Chicago, whose energy in introducing it, and in going to great trouble to modify and improve it, and adapt it to the desulphurization of copper ores and concentrates, has earned the gratitude of all copper-smelters. To Mr. W. R. Eckart, the profession is indebted for the detailed drawings of the cylinders now giving such satisfaction at the Anaconda Works, in Montana.

Through the courtesy of friends I am enabled to now give absolutely reliable information concerning the results of the enlarged and improved Brückner's cylinders now in use, having detailed information from every works in Butte City where they have been introduced.

The large cylinders, as now made, are 8 feet 6 inches in diameter by 18 feet 6 inches long.

They are lined with one thickness of good red brick, though doubtless, where fire-brick are cheap, it will pay to use them, as they withstand the mechanical wear and tear much longer than red brick; owing to the care bestowed upon their manufacture, they are much more regular in shape, thus forming a tighter and more perfect circle inside the iron shell, the strength of which can be still more increased by having the brick molded to order

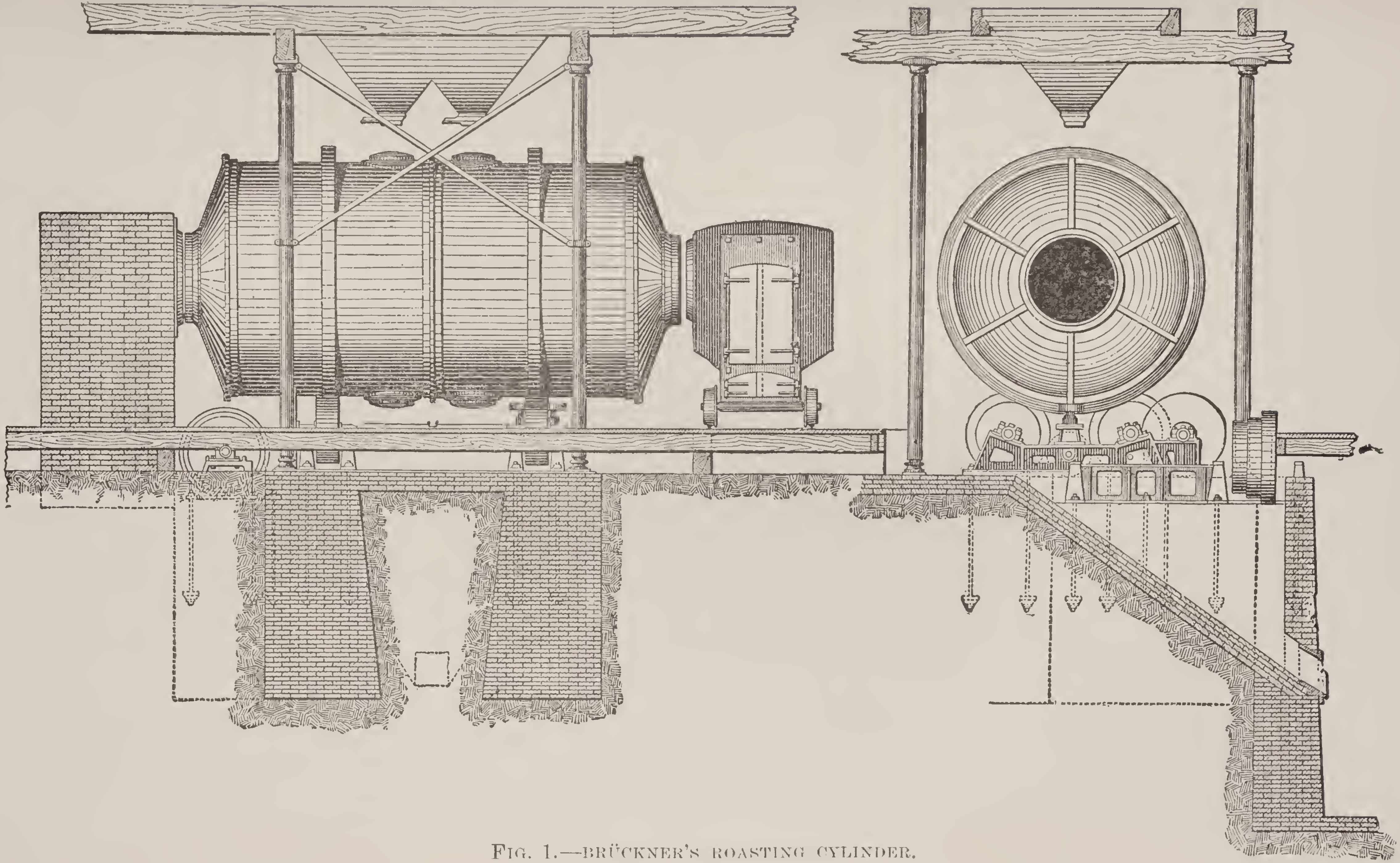


FIG. 1.—BRÜCKNER'S ROASTING CYLINDER.

to suit the inner circle of the cylinder. Again, they are much more durable when exposed to dampness than are red brick, which are quickly destroyed if dripping concentrates are fed into the red-hot furnace.

But even where red brick are used, the lining lasts about 18 months when properly put in, and as this is the principal cost of repairs during the first few years, it is evident that it must be very small.

As will be seen in the accompanying perspective sketch of the furnace, it has a double-snouted feed-hopper, with two feeding-holes, and two others opposite them, half-way around the circumference of the shell, so that it can be discharged without much loss of time. It is best, of course, where possible, to discharge the roasted ore directly into the reverberatory smelting furnaces, if such are used, or into an adjacent vault, where the heat will not be rapidly dissipated. But in works where the calcined ore must be first cooled down before going to the smelter, the cooling arrangements must be of large capacity to handle the heavy charge of ore employed.

In the improved cylinders, the fire-box is really a car, running on a track at right angles to the longitudinal direction of the cylinders, and having a short flue in one side that comes exactly opposite the throat of the furnace. In this way, the fire-box can be run opposite a cylinder, which contains a fresh charge, and fired on until the sulphur is fairly kindled. Then the movable fire-box may be wheeled along to a neighboring cylinder, and the first one left to complete the combustion of the sulphur with free access of air, and undisturbed by the reducing gases that pass through an ordinary grate. After the combustion of the sulphur, it is necessary for a perfect roast to again connect the fire-box with the cylinder and supply a little extraneous heat to complete the decomposition of the sulphates.

It is estimated that two horse power are required to drive a charge cylinder at average speed. The size and weight of the ore-charge varies greatly with its quality, percentage of sulphur, specific gravity, etc.

At the Anaconda Smelting Works, one hundred and fifty six [156] of these cylinders are constantly in operation, engaged principally in desulphurizing concentrates containing about 35 per cent. of sulphur. Here the charge is 18,000 pounds of concentrates, which in 24 hours are roasted down to 10 per cent. of sulphur. It has been found by experience that, in 36 hours, the sulphur is reduced to 3 per cent., which would be far too low for the subsequent smelting operation.

This must certainly be the largest automatic calcining plant in the world, as some 1,500 tons of ore are roasted every 24 hours.

It takes 2,000 pounds daily of Rock Springs coal to fire each cylinder. This is very inferior in quality to Eastern coal, and it would be safe to estimate that 1,500 pounds of Pennsylvania coal would be sufficient for 9 tons of ore.

As two men per 24 hours will attend three furnaces, it is evident that the cost of calcination is reduced to a lower figure by the use of these cylinders than has ever hitherto been reached.

Other smelting works in Butte, Montana, report almost identically similar results from the use of these cylinders, Mr. W. A. Clarke reporting that he is roasting a charge of 24,000 pounds down to 12 per cent. of sulphur in 30 hours, the ore consisting of three-fourths concentrates, carrying 40 per cent. sulphur, and one-fourth screenings containing 25 per cent. zinc-blende and 50 per cent. silica, using one cord of pine wood every 8 hours.

The loss of silver [as most of the Butte copper ores carry a paying quantity of silver] is not apparent, and if sufficiently extensive dust-chambers are provided, there is very little lost in flue-dust.

The capacity of any form of calcining furnace, running on pulverized material, depends upon its draught. The more cubic feet of air you can pass through the furnace per minute, the more rapidly will the process of oxidation proceed. Of course there must be a limit beyond which any excess of air will only be a detriment, but I have never yet reached that point in my own practice, nor seen it reached.

I agree entirely with Mr. R. P. Rothwell, who, having had much experience in the roasting of auriferous, arsenical pyrites in cylinders, suggests the employment of a fan to produce a proper and measured draught. He also insists on the importance of making the first dust-chamber in the series (the one next to the furnaces) very large and easy of access, by which plan almost all of the dust will settle in it, and can be easily removed without interfering with the more distant ones for many months.

The size of the grains of ore to be roasted has a most important effect upon the rapidity and perfection of the process. Of course no sensible person would use chunks of ore the size of a hen's egg (as I have more than once seen tried by inexperienced persons), as they not only tend to rapidly destroy the brick lining of the cylinder, but also retard the process of oxidation almost indefinitely.

The only exception to this rule is in the case of certain ores containing much iron-pyrites or zinc-blende, intergrown with the particles of rock in such a manner that they begin to decrepitate before a red heat is reached, and soon fly into a granular powder as completely as though they had passed through a proper crusher, and with the formation of much less dust.

I need hardly say that, in erecting a cylinder, or indeed any machinery, great attention should be paid to the foundations, which should have a solid base, with the rock or brick-work laid in thoroughly good cement mortar, and allowed to set before the timbers are placed upon it. These should be framed with the greatest care, long bolts passing down through channels in the masonry, provided with a heavy washer below, and arranged so that a long wrench can be introduced to hold the bottom nut. If the bearings are not absolutely true and level and in line at the start, and placed so that they will remain so, or can at least be easily lined-up, there will be endless trouble and wear on the running parts of the cylinder, besides a great waste of power, amounting, in certain instances that I have seen, to an increase of 100 per cent. or more in the fuel used to drive it.

These cylinders are usually arranged with fast and slow speed, so that, without altering the speed of the engine, the cylinder may be revolved slowly during the roasting period, and more rapidly whilst the ore is discharged.

A clutch of a peculiar construction is provided, so that they can be thrown in and out of gear with little effort, and with no undue jerking, or strain on the gear wheels.

By a careful comparison of the results furnished me by those who use these cylinders with the large reverberatory calciners, described at length in this book, I find that, under the conditions of high wages and expensive fuel obtaining at Butte City, Montana, the saving in cost by using the calciners instead of the reverberatories is about 40 per cent.

Where wages are low, this saving would be greatly reduced, and under some conditions that prevail in the United States, I think that, on the whole, the ordinary calciner might be equally economical. But there can be no doubt of the economy of this cylinder in the mining regions of the West.

By the courtesy of the Chicago Iron Works, I am enabled to present some excellent drawings, showing the details of the Brückner cylinders manufactured by them, and which are doing very satisfactory work in Montana and elsewhere.

Direct statements from those who are using them show that my estimate of a saving of 40 per cent. of the costs of calcining by using these large cylinders in lieu of even the most modern hand-calcining furnaces, is by no means excessive, and in some instances does not represent the full amount saved.

III. AUTOMATIC HEARTH FURNACES.

a. With stationary hearth.—This subdivision is best represented by O'Hara's mechanical furnace, which differs only from an ordinary reverberatory calciner by being provided with an automatic stirring apparatus. This consists of two or more endless chains, to which are fastened at regular intervals plow-shaped scrapers, which traverse the long, narrow hearth longitudinally, thus stirring the ore and constantly presenting fresh surfaces to the action of the air. It is looked upon with favor in Butte City, and is preferred by the Butte & Boston Copper Co. to all others.

The Spence automatic desulphurizer is a promising member of the same division, and is in successful operation at some of the acid-works near New York where copper-bearing iron pyrites are used.

The space occupied by a Spence furnace is about 34 feet by 18 feet. When two double furnaces are coupled together and run by one engine (as preferred in all cases), the space required is 34 feet by 32 feet. A building 40 feet by 40 feet is therefore necessary to accommodate this plant with a shed-roof, if connection is made to towers and chambers, or an ordinary flat-roof building with supporting posts placed between the furnaces, when connected directly with the chimney, as in the process of desulphurizing gold ores.

There are several practical points of excellence about the furnace that entitle it to careful examination by engineers. The action of the furnace will be understood to be automatic, the ores being elevated from the furnace-floor, brought in from the floor above, or by other means supplied in quantities as required to keep the hoppers full. This matter of detail will readily be understood by those practiced in the handling of ores from different levels, and the drying of the ores (if wet) will also be understood to be a simple matter when small quantities are regularly fed.

The movement of the rack (with rakes inside the furnace) opens the ports for the admission of fresh ore from the hoppers to the first shelf, and the discharge of finished or calcined ore from the lower shelf into cars. When the rakes have finished the forward stroke, the engines reverse automatically, and the rack returns to, and stops in, position.

This automatic and regular method of feed and treatment of the ore on the bed of the furnace is the result of years of study and practice, directed to the object of replacing by a uniform mechanical procedure the discretionary operation of hand labor.

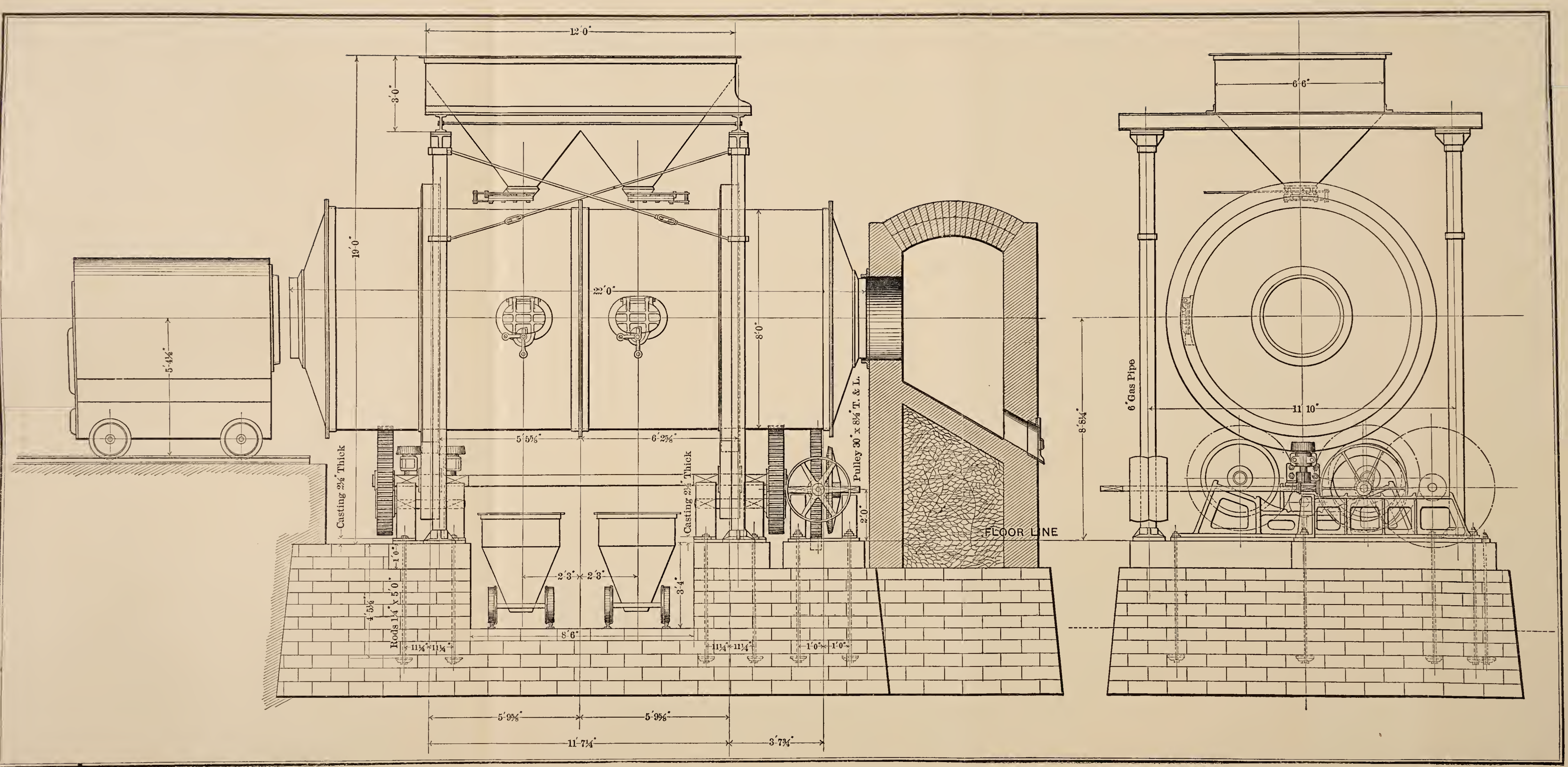
By study of the plant now in operation, the following conclusions are reached:

1. The constituent elements of the ores being first determined, the feed and discharge are regulated to exact amounts in pounds, and the number of charges fed into the furnace is duly registered.

2. The auxiliary engine being set to start the motive power, say every five minutes, and the time required for the forward and back stroke being, say, one and a half minutes, it follows that the interior parts of the rakes are exposed to action of heat and acid fumes but one-third of the time, thus approximating manual labor in wear and tear of plant.

3. The draught of air being regulated and controlled by the chemist at will, insuring the proper oxidation of the ores, and no more, less chamber space must be required than by any other process of burning pyrites, and, moreover, no special care need be given to location of plant, since strong winds or variable currents can have no effect in causing "blow-outs" of gas at the doors.

4. The movement of the ores from the hoppers to the discharge-opening is accomplished by a system of reversed teeth, which are positive in action.



IMPROVED BRUECKNER'S CYLINDER, AS USED FOR ROASTING COPPER ORES.
Side Elevation.

Front Elevation.
(Fire-box removed.)

The deterioration or destruction of cast-iron rakes and teeth has been reduced to a minimum by the simple but novel idea of *burying the parts in ore*, which accumulates at the front of the furnace-beds when the rakes are at the position of rest.

5. Pyrites "smalls," such as are found in Virginia, and at the Milan and Capelton mines, carrying 47, 45, and 40 per cent. of sulphur respectively, can be calcined with two double Spence furnaces, run by one engine at the rate of from 15,000 to 20,000 pounds per day of twenty-four hours, the cinders containing from 1½ to 2½ per cent. of sulphur.

It is claimed that larger amounts of "smalls" containing copper, blende, etc., can be put through, and double the above quantity, where sulphur fumes are passed directly into the air—as would be the case in working auriferous concentrates.

6. Where necessity exists for bringing the sulphur contents of cinders from iron pyrites (FeS_2) down to ¼ to ½ per cent. to utilize the iron, or for the like treatment of rich gold-bearing sulphurets, the result is accomplished by the addition of a fireplace to the lower hearth.

7. The average cost of calcining ores by this automatic furnace is not greater than by any other method at present in use.

The cost of the furnace, complete, with power, is about the same as that of the equivalent grate-bar space in kilns, or equal burning space in the present type of shelf-furnaces.

The following furnaces are in constant use:

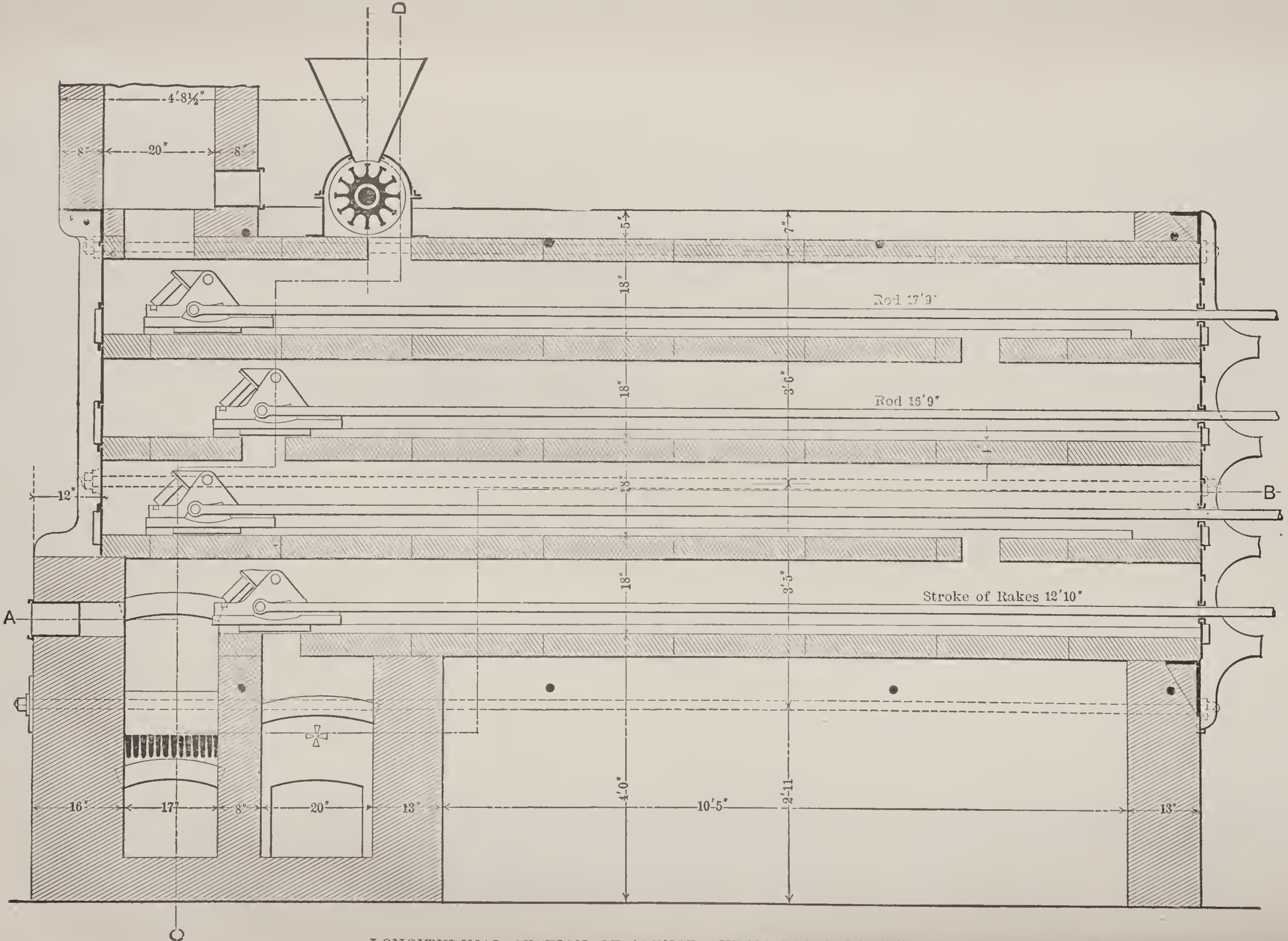
Grasselli Chemical Works at Cleveland	10
Beaver Falls	4
Titusville	3
Olean	3
Tremley's Point	6
Bergenport Chemical Works	5
Wilmington, Delaware	6
Baltimore	10
Richmond, Va.	4
Savannah, Ga.	4
Total	55

and others now building.

There are four of the latest type furnaces now working at the Parrot Copper Company's plant, Butte, Montana. The average daily capacity of the furnaces running strictly for the manufacture of sulphuric acid is about 7,500 pounds of 45 per cent. sulphur ores. The furnaces at Butte are running on a low-grade copper pyrite not to exceed 35 per cent. sulphur, and the capacity is 14,000 pounds per day. A few of these furnaces have been built with five hearths instead of four hearths shown in the illustration, the benefit being solely the reduction of the amount of sulphur in the cinders to less than 1 per cent., but no corresponding increase in tonnage of ore.

There are running at Bridgeport, Connecticut, one double furnace; at Gowanus Bay, Long Island, two double furnaces; at Elizabethport, New Jersey, two double furnaces; and others are building.

The accompanying drawing shows a Hammond improved Spence furnace used at the great Treadwell mill, Douglas Island, Alaska, where a number of them are employed in roasting the gold-bearing concentrates for treatment by the Plattner chlorination process; six double furnaces roast from 18 to 20 tons of concentrates a day to a "dead roast," with an expenditure of about $\frac{1}{8}$ cord of wood per ton of ore. If the furnaces are used as pyrites burners or if the ore contains over 35 per cent. of sulphur, and 1 to 3 per cent. sulphur in the roasted product is not injurious, no fuel is necessary. The capacity in such cases is more than double the figures given above. An expert who has witnessed the working of these furnaces at the Treadwell mill says, "This furnace may take the place of the old reverberatory, for it has all of its advantages and none of its drawbacks." The space required is small and no skilled labor is necessary. Once adjusted, it will continually discharge a finished product. Two men on a shift can attend to six double furnaces easily. One keeps the hoppers full while the other keeps the temperature even. The fronts and backs of the furnaces are so arranged that the supply of ore can be regulated exactly. The dust is even less than in the old reverberatory. A substantial hydraulic cylinder moves the rakes, which are so arranged as to prevent the banking of the material at the ends of the furnace. The iron rails of the Spence furnace, which gave much trouble, are replaced in this by very hard brick tiles. With ordinary care the iron rakes will last six months when salt is used in roasting, and two years when it is not employed, and when burnt out can be replaced by new rakes in ten minutes.



LONGITUDINAL SECTION OF SPENCE AUTOMATIC CALCINER *b. With movable hearth.*—In this type of furnace, the slowly revolving hearth is of a circular and conical shape, and may consist of two or more stories, a series of stationary rakes being fastened above each in such a manner that the ore-charge is thoroughly stirred at frequent intervals.

The Parke furnace and the Brunton calciner are perhaps the best known specimens of this type, and have been somewhat extensively used in England for desulphurizing purposes. Their great weight, cost of construction, and heavy repairs must certainly go far toward counterbalancing the slight saving in labor claimed.

IV. REVERBERATORY CALCINERS.

a. With open hearth.—This division includes virtually all of the calciners in every-day use in this country for the calcination of copper-bearing sulphides where neither the manufacture of sulphuric acid nor other outside issue has influenced the choice of apparatus.

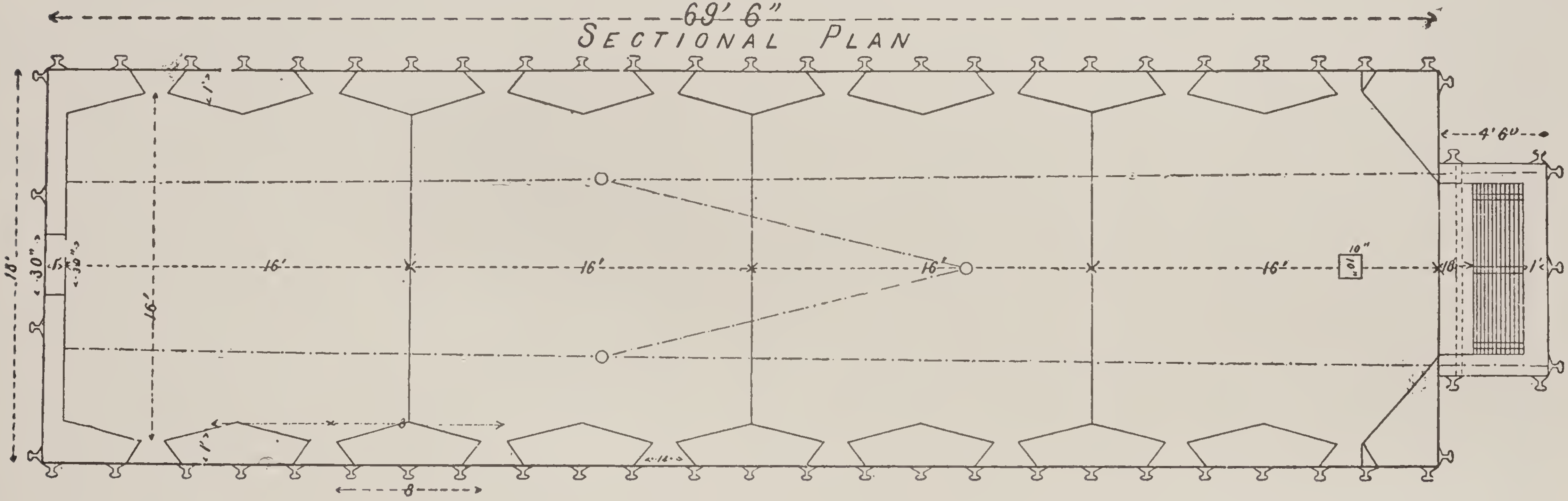
The essential features of the ordinary reverberatory calciner are a hearth, heated by a fire-place, from which it is ordinarily separated by the bridge-wall, and accessible by certain openings through the side walls, the whole being covered by a flat arch against which the flame *reverberates* in its passage from the grate to the flue, thus being brought momentarily in contact with the ore spread upon the hearth, while the combined gases from fuel and charge pass into the open air through a chimney, in many cases first traversing a series of flues and chambers for the purpose of retaining such particles of metal as may have been either chemically or mechanically borne away by the rapid draught.

A very small grate surface, as compared with the hearth area, distinguishes this type from the reverberatory smelting-furnace, and corresponds to the very moderate temperature suited to the process of calcination, permitting its almost entire construction of common red brick.

A single detailed account of the longest and largest variety of calciner in common use will serve as a model for all smaller specimens of the same class.

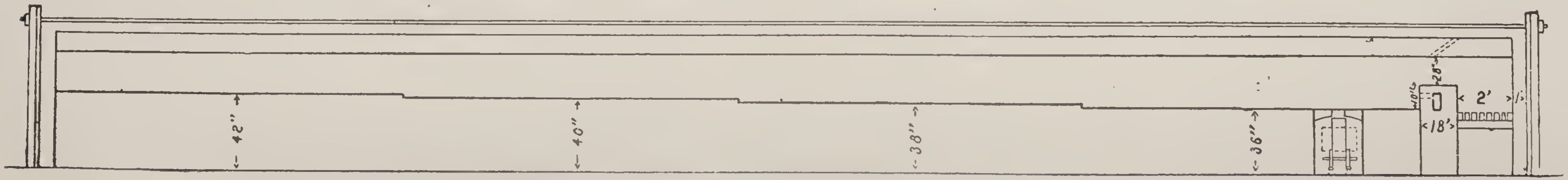
The principal variable dimension of a copper-desulphurizing furnace is its length, as, for economical reasons, its width should always be as great as is compatible with convenient manipulation. Experience has placed this limit at 16 feet for the inside

FOUR HEARTH CALCINING FURNACE



SECTIONAL PLAN

LONGITUDINAL SECTION



measurement of the hearth, nor should this dimension be lessened without good and sufficient reasons.

The length of the hearth is limited chiefly by the capacity of the ore to generate heat during its oxidation, the immediate influence of the fire-place being seldom capable of maintaining the requisite temperature upon a hearth over 16 feet in length, without resorting to the use of a forced blast, or of a draught so powerful as greatly to increase the loss in dust as well as the consumption of fuel.

The importance of the heat generated by the oxidation of sulphides in maintaining a proper temperature, and especially in conveying the heat to a great distance from the initial point, is seldom fully realized. Its intensity and durability depend upon the percentage of sulphur in the ore, and also not a little upon the manner in which it is chemically combined, the bisulphides —such as iron pyrites—furnishing a much greater amount of heat than monosulphides containing an equal gross amount of sulphur.

An ore carrying less than 10 per cent. of sulphur will not furnish sufficient heat to warrant the addition of a second hearth to the first 16 feet, which will be assumed as the normal length of a single hearth. (Such a condition would scarcely occur in practice, as, under ordinary circumstances, any copper ore containing such a low percentage of sulphur could be smelted raw.) An increase of sulphur to 15 per cent., however, will be sufficient to heat the second hearth, while a 20 per cent. sulphur ore should work rapidly in a three-hearth furnace. The addition of a fourth and final section is rendered justifiable by the increase of the average sulphur contents of the ore to 25 per cent., and even a 20 per cent. bisulphide charge may be worked to advantage in the same.

The adoption of this method of roasting, by which the ore is fed into one end of the furnace, and gradually moved to the other extremity before discharging, is attended with several obvious advantages, among which are: The gradual elevation of temperature from a point compatible with the easy fusibility of the unaltered sulphides to that degree necessary for the complete decomposition of the pertinacious basic sulphates of copper and zinc; the great saving in fuel effected by thus obtaining the full benefit of the heat generated in the process of roasting itself; the certainty that the charge must undergo a certain number of thorough stirrings and turnings in its transportation over so extended a space; the establishment of a fixed duty, which must be performed by the workmen, whose labor can thus be much more easily controlled than with the single-hearthed type of calciner, where the attendants can easily substitute an idle scratching for the vigorous manipulation necessary to move the ore forward promptly; a great simplification in firing, it being only necessary in the long furnace to maintain an even, high temperature, while with the single hearth, much experience and judgment are required to adapt the heat to the ever-varying condition of the charge; lastly, a decided economy in construction, the ratio of fire-brick to common red brick for an equal capacity of plant being much less in the employment of long furnaces.

As there seems to be almost no limit to the extent of surface over which the requisite temperature may be obtained in the calcination of highly sulphureted ores, it is very natural that experiments should have been made with still longer furnaces than any yet mentioned, 120 feet being the extreme inside length yet attempted, so far as known to the writer; but careful and repeated trials have shown beyond a doubt that no sufficient advantage is reaped to pay the increased cost of the inclosing building and other expenses of plant. It is not possible for two attendants properly to manage a furnace having more than four full-sized hearths, if the latter is pushed to its full capacity, while the addition of a fifth hearth demands a third laborer, whose time, however, will not be fully occupied, while a sixth hearth will overtax the three workmen. In short, the testimony of many excellent metallurgists, to which the author can add his own experience, unequivocally condemns the lengthening of ordinary calcining-furnaces beyond the limits above indicated, excepting under special and peculiar conditions.

The number of working-doors to a long calcining-furnace, where the ore is moved from rear to front, should be as few as possible. The limit for comfortable work should not exceed eight feet between centers of doors, and any distance less than six feet is a decided disadvantage.

The sides of the working-door frames should have short lugs, not exceeding six inches in length, cast on them, in order that they may be firmly held in position by the buckstaves, which are placed in pairs for this purpose, a single buckstaff being placed in the center of the space between each pair. The bottom of the door-frames should be on a level with the hearth surface, which should be three feet above the floor grade of the building, which should slope gradually upward toward the rear of the furnace, to correspond with the increased height of each succeeding hearth.

The common practice of filling up the portions of the hearth between the working-doors with projecting, triangular masses of brick-work cannot be recommended, as valuable space is often sacrificed in this manner. Slight projections, as shown in the cut on page 147, may be built to fill the absolutely inaccessible angles; but with properly constructed door-frames, and careful manipulation on the part of the roasting attendants, but little waste area should exist, and this will regulate itself by becoming filled with ore, which may remain there permanently. This refers, of course, to the treatment of large quantities of low-grade ores, where slight inaccuracies resulting from the trifling mixture of ores can do no harm.

After raising the side walls to the height required by the iron door-frames, usually about ten inches above the hearth level, the skewback for the main arch should be laid. This applies to the entire furnace from the beginning of the fire-box to the extremity of the rear hearth, and is a very simple matter, especially if the arch is to be perfectly horizontal, as is to be recommended in most cases. A taut line should be stretched, to insure accurate work, and if red brick are used, they should be cut on one long edge, being laid, of course, longitudinally and on the flat. They should be cut at an angle slightly greater than required by the curve of the arch, which should rise about three-quarters of an inch to the foot, making a sixteen-foot arch twelve inches higher in the center than at the sides. This rise, though less than is often recommended, will be found ample to insure perfect safety and durability, and will tend to spread the flame and heat toward the sides of the hearth.

If so-called "side skewback" fire-brick are within reach, they should be used in place of the red brick, saving much cutting and insuring a better job. Three rows, in height, of red brick, or two of fire-brick, will give a solid bearing, the total number required for a furnace of the size under consideration being respectively 600 and 375.

It is of sufficient importance to bear repetition, that all portions of the mason work above the hearth line, or wherever exposed to heat, must be laid in clay—common brick clay, tempered with sand, being quite good enough for all portions of the furnace—as fire-clay is usually expensive in the localities where copper ores abound.

Lime mortar, much improved by the admixture of a little good cement—say 10 per cent.—may be advantageously employed for the outside work, and wherever there is no danger of heat, as it makes handsomer and stronger work, and is greatly preferred by the masons, who require constant supervision to compel them to use clay mortar where it is necessary.

The heavy iron bridge plate, so indispensable in reverberatory smelting-furnaces, may be entirely omitted, the bridge being built up solid and covered on the top and sides with fire-brick, with the exception of a longitudinal opening 3 by 8 inches, which should penetrate it from one end to the other, communicating with the outside air on each side of the furnace, and with the hearth by some half-dozen 2 by 4-inch openings.

By this means, heated air free from all reducing gases is admitted into the furnace below the sheet of flame that sweeps over the top of the bridge. The oxidizing effect of this current of air is very powerful, and, as frequently determined by experiment, hastens materially the calcining process.

If wood is used as a fuel, an additional row of similar openings should be constructed in the arch, immediately over the front line of the bridge-wall, by which a much more perfect combustion of the gases is effected. With coal as a fuel, the latter openings are superfluous, provided the firing is properly managed.

Aside from the sixteen working-door frames, and the ordinary doors for fire-box and ash-pit, no castings are necessary for the entire structure, excepting a small frame to protect the charging-hole, which should be situated a little back of the center of the rear hearth, being placed, of course, in the medium longitudinal line of the furnace. It will add also materially to the durability of the fire-box to surround the portions of the same most exposed to pressure or mechanical violence by light cast plates, held in place by the uprights.

As the portion of the hearth immediately beneath the charging-hole is exposed to excessive wear from the constant precipitation of heavy masses of wet material upon it, an area some three feet square, and in the locality designated, should be constructed of either fire-brick or slag blocks, the latter, from their texture and general physical condition, being peculiarly well suited to the purpose.

By referring to the sketch on page 147, it can be plainly seen at what stage in construction the various bearing bars and other iron work must be inserted.

It will be noticed that, instead of adopting the ordinary large ash-pit, entirely open at the rear, according to the invariable English practice, preference is given to a closed ash-pit, to which air is admitted by a door at one or both ends. This effects a great saving in fuel, and brings the process of combustion more perfectly under control. Comparative tests, extending over a considerable period of time, show this saving to amount to about 50 per cent. of the entire fuel consumed, in the case of coal, and about 65 per cent. (in volume) when pine wood is used. The tight ash-pit becomes, of course, a matter of positive necessity where anthracite coal, with a forced blast, is used.

The side and end walls having been carried up to the required height, and the skewback constructed on both sides for their entire length, the carpenters take possession temporarily, usually under the supervision of the head mason, to put in the wooden center on which the arch is to be built. If a second furnace, or indeed any other work, is available for the remaining masons, it is advantageous, though not indispensable, to permit the furnace to stand uncovered for several days, thus allowing the mortar to set, and greatly increasing the strength of the mason work.

Having selected for description that pattern of calciner in which the gradual diminution of the space between arch and hearth, as it recedes from the grate, is due to successive slight elevations of the hearth level, instead of the ordinary downward pitch of the roof, it is evident that the arch throughout its entire extent will be horizontal, while all four inclosing walls are built up to the same height at every point, with the exception of a rectangular flue-opening in the rear wall, 6 by 30 inches.

The construction of the wooden pattern or center is, therefore, extremely simple, requiring only some 20 pieces of 2-inch plank, 16 feet long and 14 inches wide; a lot of 2 by 4 scantling, to form posts about 10 inches in length, four of these being needed to support each plank on edge; and finally, a sufficient amount of 4-inch battens, from one-half to one inch thick, to cover the area of the required roof, when placed about three-quarters of an inch apart. The plank should be perfectly sound, and at least partially seasoned.

By the aid of a long rod, moving upon a pivot at one end, while the free extremity carries a pencil, a segment of a circle corresponding to a rise of 12 inches in the center of the length of 16 feet, should be struck on each plank, and the line followed accurately with a jig-saw.

The segments for that portion of the arch over the bridge and fire-box are shorter, of course, than those belonging to the main hearth, but should be got out in the same manner, and then shortened at each end to the required length.

The scantling should be cut into posts somewhat shorter than necessary to bring the curve on the upper edge of the segments to the proper height for the lower surface of the arch, so that each post may be wedged to an exact bearing with thin slips of wood. It is quite necessary that the weight should be evenly distributed, and each segment, when brought into correct position, is held there by driving a nail through a longitudinal line of battens in the center and at each extremity.

The segments for sloping arches should be still further strengthened by short braces toe-nailed obliquely from the upper edge of one strip to the lower edge of its neighbor, and so on throughout the entire frame.

An omission of this precaution has caused the canting of the segments and consequent destruction of a large, nearly completed arch under the author's charge.

No difficulty will be experienced in removing the wooden pattern in good condition for further use, provided it is supported on small posts as just described; but if long, heavy blocks of timber are used as a foundation for the segments, great labor as well as much injurious sledging must accompany their removal, resulting usually in the complete destruction of both segments and battens. In fact, where this method of support has been practiced, it will be found best to burn out the inclosed patterns, after the tie-rods are properly tightened, closing both damper and ash-pit so as to allow only a slow smoldering, and prevent any injurious rise of temperature in the still damp furnace.

Few jobs of mason work require more care and conscientiousness than the laying of a large calciner arch, as, owing to its great width and slight curvature, a very little lack of closeness in its myriad joints would be sufficient to allow it to yield to the enormous pressure brought to bear by its own weight, and become sufficiently compressed to slip down between its side walls. It is quite a simple matter to lay a good solid arch of fire-brick, owing to their great regularity and smoothness and almost perfect rectangular form; but when red brick are used, which vary so in size and thickness, and are so frequently warped out of all reasonable shape, much care is required.

In ordinary calciners, it is customary to construct that portion of the arch from the fire end of the furnace to a point midway between the first and second working-doors of fire-brick, nine inches in thickness, the brick standing endwise. At this point, or even considerably sooner, when necessary, red brick are substituted, being placed also on end, each brick, after being dipped into a pail of liquid clay mortar, being pressed closely against its neighbor, and finally settled into position with a few light blows of the hammer.

Moderately soft brick are, as a rule, best suited to this purpose, although they must, of course, possess ample solidity to resist the compression to which they are exposed. Hard-burned brick, though stronger, are too irregular and warped to be often used in a large arch, and in any case the brick should be all carefully selected beforehand by the attendant, and assorted in such a manner that each longitudinal row—extending the entire length of the furnace—is composed of brick of about the same thickness.

Another most important precaution is the preservation of the proper angle, as, in order to establish the required curve, each row must incline slightly from the vertical, the lower end of the bricks being in contact, which is not the case with their upper extremities.

The establishment and preservation of the proper curvature are facilitated by the occasional interpolation of a longitudinal row of wedge-shaped or key-brick, technically called "bullheads." These are usually only obtainable made from fire-clay, but are almost indispensable for the center row, when the final keying of the arch is effected. Otherwise, the entire row of key-brick must be cut from common brick, an arduous and imperfect task.

The keying is a matter of some delicacy, and should be performed by a single workman, who should select or cut his keys of such thickness as to produce a uniform moderate pressure throughout the entire distance, no more force being exerted to drive the key into place than can be easily effected by a light mason's hammer, using an intervening block of wood to prevent the destruction of the brick.

While the masons are thus employed, the blacksmith and his helper should have completed the buckstaves and tie-rods from measurements furnished by the foreman mason as the work progresses, it being in such cases easier to suit the length of the tie-rods to the completed mason work, than to pursue the opposite course.

As soon as the arch is completed, the head mason and blacksmith should proceed to the ironing of the furnace, which, with the assistance of two laborers, should be completed in a single day.

The most convenient and easily obtained buckstaves in most cases are old iron rails of full size, say, 80 pounds to the yard. Properly shaped I-beams, of corresponding strength, are about 15 per cent. lighter. The tie-rods may consist of one inch round-iron for the bottom rods, and one inch and a quarter iron for the upper rods. The lower rods are already long in place, and through each of their loops should now be slipped one of the upright buckstaves, cut to the proper length, and temporarily wedged into the loop to keep it perpendicular.

The upper tie-rods may be made the same as the lower, with a loop at each end—the necessary tightening being effected by flat-iron wedges; or they may have a threaded extremity at one end passing through a corresponding hole in the buckstaff, and fitted with a strong nut; or, best of all, a small ring is formed at one end of the tie-rod, through which slips a U-shaped piece of round iron, which fits against the buckstaff, on the other side of which a piece of flat iron, pierced with two holes for the free ends of the U, is held, these ends being threaded; a nut for each of the ends completes the apparatus, and presses the piece of flat iron tight against the upright. This is a simple and highly satisfactory device, and avoids the disagreeable process of wedging in the one case, or of punching a large hole through a narrow rail in the other. The strain is distributed over two bolts and nuts, and can be instantaneously increased or diminished; nor will the nuts rust solid into place, provided they are saturated with oil annually, and slightly turned, to free them.

Whatever method of tightening the tie-rods may be selected, the process of ironing or anchoring should begin with the first tie-rod on the *main body* of the furnace, nearest the fire end, and proceed systematically toward the rear, thence returning to the shorter transverse rods that support the arch over the grate, and terminating with the long longitudinal rods, which, for convenience of handling, should be in three lengths, connected with hooks and eyes. Up to this time, no great strain should be put upon the rods, everything being merely brought to a solid bearing; but after all are in place, and the buckstaves evened both vertically and laterally, the rods may be drawn to the desired tension, the skewback being still further supported by a bar of one by four inch flat iron, or, better, an iron or steel rail, let in flush with the brick-work.

This is largely a matter of experience, and being of vital importance, should receive the most careful attention on the part of the builder, as too lax a condition of the rods may permit the entire falling in of the arch, while the contrary fault may cause a positive buckling and elevation of the same, accompanied with a general cracking and distortion of the lateral walls. The latter accident, in a moderate degree, is much more likely to occur than the former, owing to the natural tendency to overdo a measure essential to safety, and yet not exactly defined.

The lateral rods should be tightened until they begin, when struck near the center with a hammer, to vibrate rapidly, and to be but little depressed when stepped upon. (It is almost needless to say that none of the upper rods should touch the arch.) A simultaneous examination of the brick-work forming the upper portion of the side walls should also be made, as it is here that the effect of the curving of the buckstaff from too great tension, and consequent pressure against the mason-work, is first visible.

The extreme limit of tension is reached when the first signs of this appear, as nothing can be gained by bending the uprights, and if the latter are sufficiently strong and applied in the numbers shown in the illustration, the arch may be considered perfectly supported. All the rods should be tightened to about the same extent, although it must be remembered that the great length of the longitudinal rods may prove deceptive in estimating their tension, it being impossible to tighten them to such a degree as the shorter lateral ones.

A single additional precaution is recommended, though seldom practiced by builders. This consists in breaking up a few thin roofing slates into fragments a couple of inches in length, and driving these with moderate force into whatever crevices may still be found in the surface of the arch.

Some twenty or thirty pails of liquid mud are now poured over the arch, and the process repeated as it dries, until every crack and crevice is filled, and the roof rendered completely solid and air-tight.

The wooden center on which the arch was built should now be removed by first knocking away the little posts that support it, using a light stick of timber as a battering-ram, and proceeding from one side door to the next until every stick and batten are removed. They should be stored for future use. Any indications of settling on the part of the arch must be immediately counteracted by tightening the tie-rods; but when the precautions enumerated have been carefully observed, this can never occur.

The length of time the completed furnace may now stand untouched with advantage to the mason-work is only limited by the requirements of the business, which almost invariably require its being put in commission at the earliest possible moment. Under such circumstances, a smoldering fire of large logs, knots, or any slow-burning waste material, should at first be kindled on the floor of the ash-pit, the grate-bars not being put in place until the masonry surrounding the fire-place is partially dried.

In twelve or eighteen hours, the fire is elevated to its proper place, and with a nearly closed ash-pit door and partially lowered damper, the process of drying proceeds gently and without that violent generation of steam and vapor that is sure to be accompanied by extensive fissuring of the brick-work and permanent weakening of the entire structure.

A most careful and repeated examination of the condition of tie-rods and buckstaves should be made every twelve hours from the first kindling of the fire until the furnace has attained its full heat, and may be supposed to have expanded to its utmost limits, although it may be a month or more before all evidences of movement cease. The first indication of this process will be seen in the neighborhood of the bridge and fire-place, where the highest temperature prevails. A bending of the buckstaves, combined with a pressing in of the skewback line and an increased tension of the cross-rods, are warnings that may soon be followed by either a complete giving way of some portion of the iron-work, or more frequently by a bodily upheaval of the arch and general fissuring of the brick-work, unless relieved by diminishing the strain to a corresponding degree. This process of loosening must be extended to the entire iron-work of the furnace, and continued as long as necessary, the tension being again increased if the furnace is ever allowed to cool down to any considerable degree—an operation more destructive to it than many months of ordinary wear.

While the apparatus is thus gradually being brought into proper heat, the sheet-iron hopper should be suspended from timbers resting upon the trussed beams of the building. It should be strongly constructed and well braced, and provided with a stout lever, easily accessible to the operator when standing upon the floor of the building. A track running transversely to the row of calcining-furnaces, and parallel with the longitudinal axis of the building, renders these hoppers easily accessible to the car in which each weighed charge of ore is brought. The car is provided with a dumping arrangement, so that it easily and completely empties itself into the furnace hopper. The laborer who weighs and transports the charges can supply six furnaces, provided everything is arranged as herein described, or in a similarly judicious manner.

The outfit of tools may also now be prepared, and should consist, for each four-hearth calciner, of 6 rabbles, 4 inches by 10 inches and 12 feet long; 6 paddles, 8 inches by 12 inches and 12 feet long; 4 door-hooks, to handle the sheet-iron working-door; 1 long-hooked and pointed iron poker for wood, or an ordinary coal poker, if coal is used; 2 ordinary long-handled, square-pointed shovels; 1 scoop-shovel (for coal).

The iron rollers, usually employed as rests for the long tools at each working-door, soon lose their shape and cease to revolve. It is better, therefore, to provide merely a smooth iron bar, which, if kept well soaped, renders the handling of the tools as easy as any of the more expensive devices.

When available, a free-burning semi-bituminous coal forms the most economical fuel for calcining purposes, but should always be burned upon a comparatively shallow grate, instead of using the deep clinker bed, so suitable to the smelting process. At the comparatively low temperature suited to calcination, the generated gas does not burn perfectly, and a great waste of fuel occurs. Coal should be fed at short intervals—from 30 to 45 minutes—in quantities seldom exceeding 50 pounds. When wood is cheap, nothing can excel it as a fuel for calcining purposes, its long, hot, non-reducing flame being peculiarly suited to the requirements of the process. About one and two-thirds cords of hard, or two cords of soft, wood are commonly considered equal to 2,240 pounds of good bituminous coal.

CONSTRUCTION OF FURNACE STACKS.

While the furnace is gradually getting into condition for its first charge of ore, an opportunity is offered to return to the question of construction again, and describe a method of building stacks that is much more economical than that usually pursued, and which, though not new or original, is certainly not generally adopted in the erection of smelting-works.

Owing, perhaps, to the influence exercised by studying the practice of most European builders, and by following former customs without thinking particularly of the possible opportunities for improvement, the vastly greater number of furnace stacks now erected are very much more costly than they need be, both as regards labor and material. From a very recent comparison of costs with professional friends, the author has found that the average calciner stack, as erected by him during the past twelve years, has cost less than one-half the amount ordinarily expended, making it worth while to occupy a few paragraphs in describing the more economical practice.

The most important feature of a chimney is its foundation; but it is at this very point that a great saving over ordinary practice may be effected without lessening the stability of the superstructure.

A mere increase in depth below the loose soil forming the surface of the ground does not add in the slightest to the value of the foundation, after a proper material for the same has once been reached; and as this occurs in the greater number of cases within three or four feet of the surface, the frequent practice of additional excavation for the apparent purpose of merely gaining depth is money thrown away.

After removing the loose surface soil, and penetrating below any danger of frost, in the greater number of cases no advantage would be gained by excavating to a depth of 50 feet, unless solid bed-rock were reached.

Any kind of gravel, hard-pan, or even soft loam or sand, if homogeneous, will answer the purpose perfectly, it being understood that reference is here made to an ordinary calciner or smelter stack not exceeding 80 feet in height.

In the case of a yielding sand bottom, and especially if the line of division between two strata of varying quality happens to cross the excavation, it is well to form a solid floor to the pit by putting in a double layer of three-inch plank, nailed cross-wise But in all ordinary cases the hole should be simply filled with broken stone, about the size of ordinary road metal. This material, when well rammed into place and thoroughly grouted, by pouring in a sufficient quantity of mortar composed of one part each of lime and cement, and three of sand, makes a foundation infinitely superior to one formed of a few large stones, the slightest settling of any one of which will throw the chimney out of perpendicular.

The excavation should be at least two feet larger in every direction than the base of the chimney, and the stone-work of the latter, laid in lime and cement, should cease some three feet below the surface, at which point the brick-work usually begins.

If a cupola smelting-furnace is in operation in the immediate vicinity, nothing can be more satisfactory or economical than the following plan, pursued by the author on several occasions:

An excavation being made of the usual size, the molten slag from the smelting-furnace is wheeled to the spot in the usual movable slag-pots, and poured at once into the hole, which, when filled to the proper height with the fused rock, and leveled by means of little clay dams along the edges, so as to present a smooth surface for the masons to begin on, will contain a solid block of lava, weighing many tons, and as immovable as a ledge of rock.

In constructing a stack, we have to determine the size of flue desired, and intimately connected with the same is the degree of *batter*, or taper, which shall be given to the structure.

The object of this batter is two-fold: 1st. For appearance. 2d. For the sake of strength. The first reason may be entirely neglected in metallurgical architecture, and experience has shown that, within the limit of height mentioned, a batter of one-eighth of an inch to the foot is ample. Nor need the taper be begun until the stack rises above the roof, as that portion of the structure within the building is amply protected from the force of the wind.

By thus decreasing the amount of taper, we greatly increase the capacity of the stack, as experience shows that a contraction of the flue in its upper portion is accompanied with a corresponding diminution of draught, while a positive enlargement of the same toward the top has a most beneficial influence. This latter point is gained by lessening the thickness of the chimney walls as they grow higher, while the outside taper remains constant.

All calculations and formulæ regarding the necessary size of any flue for a given duty have been found so greatly modified by circumstances—such as variations of internal and external temperature; humidity of atmosphere and state of barometer; change of winds, etc.—that it is found safest to rely upon experience and analogy; and after beginning with a much larger flue for safety, the author has finally found a stack 42 inches square inside, at its narrowest part, and 65 feet high, to possess ample capacity for two large calcining-furnaces such as just described. It is proper to add that a much smaller stack will produce the draught usually considered as quite sufficient for the calcining process; but long-continued experiment has shown such extraordinarily favorable results, as regards both capacity and perfection of roast, to arise from greatly increasing the ordinary calciner draught, that a sharp and powerful draught appears as essential to a calciner as to a smelting-furnace.

For this reason, also, no more than two furnaces should be led into a common stack, it being almost impossible properly to equalize the admission of air to each calciner, and to produce that sharp and vigorous draught so essential to rapid oxidation, and especially to the conveyance of the sheet of flame and heated gases over the whole length of a 4-hearth calcining-furnace. The interposition of dust-chambers, or preferably of large flues, filled with parallel rows of sheet-iron, according to the method found so efficient and economical at Ems, is of course necessary, and should be present in any case.* Limited experiments conducted by the author fully satisfy him of the great benefit to be derived from the adoption of this economical and efficient method of condensation.

The size of chimney mentioned—42 inches—will answer for all elevations up to 5,000 feet above sea-level. For each 1,000 additional height, these figures should be increased one inch.

For a calciner chimney of this size and 65 feet in height, the walls at the base should be 17 inches thick, the length of two red brick, no fire-brick being needed, as the gases are sufficiently cooled by their passage through the long furnace and flue. This thickness is maintained for a height of 25 feet from the ground, which brings it somewhat above the roof of the building. At this point, the external batter of one-eighth of an inch to the foot is begun, and an internal set-off of 4 inches is taken; thus decreasing the thickness of the walls to 13 inches, and enlarging the flue to 50 inches.

This constant taper is maintained by the employment of an ordinary beveled plumb-bob, which obviates any trouble or calculation. This condition of affairs is continued for another 25 feet, during which distance the flue is contracted to a size of about 44 inches, when another internal 4-inch set-off is taken, increasing the same to 52 inches, while the walls are diminished to 8 inches.

This, being continued for 15 feet, gives the full height of 65 feet, the flue at the top being still 48 inches square, or 6 inches larger than at the base. No ornamental finish at the top should ever be allowed, the stack either being surmounted by a light casting to hold the brick in place, or left without this protection, the iron braces being usually sufficient to prevent the loosening of the upper rows of brick-work. An ornamental cap is simply a source of annoyance and danger, and should never be permitted in a stack devoted to the passage of sulphurous vapors.

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* See description of Ems method of condensation, by Professor Egleston, in *Transactions of the American Institute of Mining Engineers*, xi., 379.

A chimney of this size is best built from the outside, a scaffold being erected by placing eight stout poles about the base of the proposed structure, nailing cross-pieces at the proper height for the plank staging, and thoroughly bracing the uprights by boards nailed diagonally from one to the other.

The uprights may be lengthened out almost indefinitely by careful splicing, and as the stack grows higher, new cross-pieces are spiked every five feet, and men and material thus maintained at the desired elevation. A rope and bucket, with a single wooden block fastened to the railing of the staging, and manipulated principally from the ground level, form the most economical means of elevating the requisite material, while a single laborer above is able to furnish four masons with brick and mortar, most of the work being done from below. It is best to employ four masons, so that one can work on each wall of the stack, and their position should be changed twice daily, in order to equalize any differences in the amount of mortar used, etc.

Like all other mason-work that is to be exposed to heated sulphurous gases, the interior portion of the stack must be laid in clay mortar (ordinary sticky mud); while the remainder of the structure should be laid in lime mortar, on account of its superior tenacity. To prevent the penetration of the vapors into the porous brick, the interior of the flue should be thoroughly plastered with clay throughout its entire extent.

While the durability of a chimney of this description is largely dependent upon its being ironed, it is still more dependent upon its not being ironed too stiffly. A stack with corners thoroughly inclosed in stiff angle iron, tightly held together with frequent braces, will fissure and give out in a few years, while a similarly built chimney containing a few light irons merely to hold the brick-work in place, will last from twenty to forty years.

This is the result of personal experience, confirmed by the observations of most other constructing engineers, and is especially the case in countries where high winds and violent fluctuations of temperature are prevalent.

Eight uprights of $\frac{3}{8}$-inch by $\frac{3}{4}$-inch iron, each upright being placed about 4 inches from each corner of the stack, and passing through rectangular openings cut in $\frac{1}{2}$ by 2-inch flat iron, which latter pieces are laid in the brick-work from 30 to 36 inches apart, are amply sufficient for the purpose. The holes must be so punched that the uprights can be wedged tightly against the brick-work, which is thus held in place even after the mortar has long succumbed to the combined influence of the roast gases and the elements. As a striking example of the accuracy of the above remarks, the reverberatory smelter stacks of the Detroit Smelting Company's copper-refining furnaces at Lake Superior may be mentioned, where, on building a strongly ironed stack, they found it fissure and become unsound in a very short time; whereas their ordinary stacks, anchored only by means of occasional straps of flat iron built into the chimney walls and bent over at each end, have stood for fifteen years or more without showing crack or imperfection.

A row of headers should be introduced at about every eighth course, and the lower portion of the stack into which the two calciner flues enter on opposite sides should be divided by a 4-inch partition wall into two equal compartments. This wall, extending some five feet above the entrance flues, serves to bend each current in an upward direction, and thus prevent the whirl and disturbance of draught resulting from the meeting of two opposing currents.

The following interesting observation has been communicated by Messrs. Cooper and Patch, superintendent and chemist of the Detroit Refining-Works:

In most reverberatory furnaces, the flue enters the stack at some distance above its base, and consequently there is a cavity inclosed by the chimney walls, of greater or less depth below the embouchure of the flue. When this apparently useless cavity has become filled up from the falling in of the stack lining, drippings from the molten brick, or other causes, the draught at once suffers and the capacity of the furnace is greatly diminished.

Whether this phenomenon arises from the loss of the elastic air-cushion that is normally present, or whether there is some other reason, the fact remains, and although the observations have been confined mostly to smelting-furnaces, it is probable that a calcining-furnace may be affected in a similar manner, and therefore, in all cases where a horizontal or inclined flue enters a stack, it should be so constructed as to leave an open space of from 4 to 6 feet below it. This need not communicate with the outside air in any way, except for the purpose of cleaning the stack or entering it for repairs.

It is well to provide every high stack with a good lightning-rod, properly fastened and insulated.

The building that covers any considerable number of calcining-furnaces is necessarily of great extent, and should, if possible, be built of very light and, at the same time, fire-proof materials.

Scarcely anything fills these requirements so thoroughly as a medium grade of corrugated iron. This, if well fastened down, and painted every three or four years, will be found the most economical and satisfactory material for both sides and roof that is yet known. If the number of furnaces under a single roof exceeds two, they should be placed at right angles to the greatest length of the building, a space of only three feet being left between the rear end of the furnace and the corresponding side of the building, while between the fire-box and the lower side of the building there should be ample room for a drive-way for the conveyance of fuel, as well as for a railroad paralled to the same and close to the wall, over which the calcined ore may conveniently be dumped into a paved and roofed inclosure on a level as low as the circumstances of the case permit. The sixteen-foot calciners should be separated by spaces of at least fourteen feet.

As the main building for these long calcining-furnaces must be from eighty to ninety feet in width, it is often the practice to support the cross-beams on posts that, if properly placed close to the furnace and midway between the working openings, need not interfere with the long tools in use. But there is no difficulty in constructing trusses to support a roof of this size without the aid of posts, nor need the expense be much greater. The principal difficulty is encountered in raising these immensely long and heavy "bents;" but this may be entirely obviated by constructing a series of cheap scaffoldings, and putting them together piece by piece, instead of attempting to raise the entire "bent" bodily. The ridge of the roof should be surmounted by a continuous ventilator throughout its entire extent. The details of this work may be intrusted to any experienced carpenter.

COST OF CONSTRUCTION OF CALCINING-FURNACE.

The following estimates of cost are taken from notes that cover the construction of a considerable number of large calcining-furnaces, and being given without alteration or omissions, excepting the necessary reduction to our assumed standard of costs, should furnish reliable figures on which to base future plans:

COST OF ONE FOUR-HEARTH CALCINER.

Excavation—45 days at $1.50	$67.50	
Removal of material excavated	35.00	
Superintendence and miscellaneous	24.00	$126.50
Foundation walls—1,840 cubic feet.		
2,000 slag-brick at 2 cents	40.00	
20 days stone-mason and helpers	120.00	
Materials for mortar	28.00	
Labor on same and utensils	16.00	
Miscellaneous labor	12.00	
Superintendence	15.00	$231.00
Brick-work on furnace proper.		
2,420 cubic feet, say 50,000 red brick at $8	$400.00	
7,500 fire-brick at $40	300.00	
Lime and sand	137.00	
4 tons fire-clay at $8	32.00	
8 tons brick-clay at $1.50	12.00	
32 loads sand at $1.50	48.00	
112 days' brick-masons' labor at $4	448.00	
112 days' ordinary labor at $1.50	168.00	
3 days' carpenters' labor at $3	9.00	
Miscellaneous labor	35.00	
8 days, blacksmith and helper	40.00	
Materials consumed by same	8.00	
Iron work.		
Superintendence	112.00	$1,749.00
66 buckstaves (old rails), 6½ feet long, 80 pounds, at 1⅝ cents per pound	$85.80	
Tie-rods and loops, 2,056 feet, 1¼-inch round iron = 8,327 pounds, at 2 cents	166.54	
Flat iron for skewback, grates, etc. = 2,064 pounds, at 2 cents	41.28	
16 cast frames and doors, at 156 pounds each = 2,496 pounds, at 2½ cents	62.40	
Fire-doors and other small castings	16.50	$372.52
Nuts and bolts		6.25
Short flue, with damper and ½ cost of stack		364.00
Grading and miscellaneous		47.50
Tracks for feed and discharge of ore		62.40
Set tools, complete, as per former schedule, 1,250 pounds, at 2 cents		25.00
Labor on same		18.00
One iron ore-car (list price)		85.00
Grand total		$3,087.17

The repairs on a thoroughly built calciner should be nothing for the first three years; for the succeeding seven years, they will average 3 per cent. per annum on its first cost, while from its tenth to its fifteenth year, 5 per cent. per annum will probably be expended in renewing the hearth and roof once and patching the furnace in various places.

After fifteen years of constant usage, it is cheaper to build a new furnace than to keep the old one in repair; but few metallurgical enterprises in this country require to provide for a period longer than the above.

The variety of reverberatory calciner known as the *muffle furnace* is now seldom used by the copper smelter, as, except for purposes of acid manufacture, it possesses few advantages above the ordinary hearth variety, and in case this branch of metallurgy is also practiced, some of the newer forms of automatic furnaces have displaced the muffle. The high cost of construction and greater consumption of fuel are also adverse to its employment, and although, from its gentle and regular heat, it possesses decided advantages in the treatment of easily fusible substances, it is rather suited to the calcination of matter containing much lead, or of pyrites with salt, as in the Henderson process, none of which operations comes within the scope of this treatise.

An easily fusible ore can be very efficiently protected from the fierce heat of the first hearth of an ordinary calciner by the construction of a four-inch curtain arch, covering one-third or more of its surface from the fire-bridge onward, though such a precaution is seldom necessary, excepting in the case of matte calcination, which requires but slight modifications of the roasting process as applied to ordinary sulphide ores.

The process of ore calcination, like most other operations based on chemical reactions, must be understood before it can be properly and intelligently executed, and no description of the same would be complete without a brief review of the chemistry of the calcining process.

CHAPTER VIII.

THE CHEMISTRY OF THE CALCINING PROCESS.

A SUFFICIENT idea of the chemical reactions that occur in this important metallurgical process may be obtained by following an ordinary pyritous ore in its passage through the roasting-furnace, and carefully noting all the changes that it undergoes from the moment of its introduction until it is ready for the succeeding fusion; nor are the conditions in either roast-heaps or stalls so different as to require any separate consideration.

A typical ore for this purpose might consist of a large proportion of pyrite, say 45 per cent., some 20 per cent. of chalcopyrite (containing about one-third copper), with a slight admixture of zinc-blende, galena, and sulphide of silver, while the remainder of the ore would usually consist of quartz or siliceous material, which may be regarded as practically inert in its effect upon the process of calcination. A charge of such ore, being introduced upon the hearth of a roasting-furnace still at a bright red heat from the preceding operation, exerts a powerfully cooling influence upon the glowing brick-work, and within ten or fifteen minutes reduces the temperature to a point below the ignition-point of sulphur, the ore at the same time giving off its moisture, and gaining so much heat that a very slight aid from the fuel on the grate is sufficient to start the oxidation of the iron pyrites, as shown by the blue, flickering flame that plays over the surface of the charge, beginning at that portion of the same that borders on the already hot charge occupying the adjoining hearth, and gradually advancing toward the rear, until every square inch of surface is in a state of active combustion. The rapidity of this process of oxidation varies according to the degree of temperature and the sharpness of the draught, but should not occupy more than an hour from the first introduction of the charge. The composition of iron pyrites (FeS_2) is such that, while one atom of sulphur is united to the iron with considerable tenacity, the second atom is held by very feeble bonds, and becoming volatile at the moderate temperature of the calcining-furnace, unites with the oxygen of the air, forming sulphurous acid (SO_2), which escapes in the form of an invisible gas. This reaction is accompanied by a very considerable evolution of heat and the flickering blue flame already mentioned. Being entirely dependent upon the oxygen derived from the air, this reaction is confined principally to the surface of the charge, which, if left undisturbed, would soon undergo a slight fusion, causing a caking of the ore, and still further hindering the extension of the process. It is therefore just at this point that the necessity for frequent and vigorous stirring becomes strikingly apparent. By this manipulation, any incipient crust that may have formed is broken up, the temperature of the layer of ore is equalized throughout its entire depth, and fresh particles of ore are constantly exposed to the influence of the air.

The stirring should begin on the first appearance of the blue flame, and continue for ten minutes at a time, with equal intervals of rest, during which time the working openings should be closed, while an ample air supply is admitted through the regular channels provided for this purpose. The stirring should take place from both sides of the furnace at the same time, and should be systematic, vigorous and thorough; extending to the very bottom of the charge, and omitting no portion of the ore.

During this period of roasting, and until the disappearance of the blue flame, the roast gases consist almost exclusively of sulphurous acid together with steam from the moisture present, and the invariable products of the combustion of the fuel.

It will, of course, be understood that the SO_2 and other roast gases form but a small proportion—seldom more than 2 per cent.—of the air issuing from a calciner stack; atmospheric air always being present in overwhelming proportions. The SO_2 results from the direct oxidation of one atom of the sulphur contents of the iron pyrites, or, when the temperature is somewhat high, of the absolute volatilization of this atom of sulphur as sulphur, and its immediate combustion to SO_2.

The next stage of the process may be reckoned from the beginning of the oxidation of the iron of the pyrites, and also of its second atom of sulphur. This is a much less rapid and vigorous process than the preceding, and is attended by the formation of a certain amount of sulphuric acid, in addition to the sulphurous acid, which is still generated in large quantities. The means by which the former acid was produced was not clearly understood until Plattner's patient and ingenious researches developed the "contact theory," according to which sulphurous acid and the oxygen of the air, in the presence of large quantities of heated quartz, or other neutral material, combine to form sulphuric acid, which may escape invisible, or in the form of white vapors when hydrated, or may in the instant of its formation combine with any strong base that may be present.

In the case under consideration, protoxide of iron (FeO), arising perhaps from the very particle of pyrites whose oxidation gave rise to the sulphuric acid, is at hand; and while the greater proportion of the sulphuric acid formed escapes into the atmosphere, a certain amount combines with the protoxide of iron to form ferrous sulphate, whose presence may easily be detected, owing to its solubility in water.

From the very commencement of the formation of sulphuric acid, a new and powerful oxidizing agent is gained, as the protosulphate of iron is easily broken up by heat. The decomposition of its acid into SO_2 and O promotes the oxidation of other sulphides present to sulphates, while the protoxide of iron is raised to the sesquioxide of that metal—a tolerably stable compound, and one usually found in large quantities in thoroughly roasted pyritic ores. Before the complete decomposition of the ferrous sulphate has occurred, and indeed while some considerable proportion of sulphide of iron may yet remain, an analogous process takes place with the chalcopyrite, its ferruginous portion following almost precisely the same course as the iron pyrites, while its copper contents are transformed into cupric sulphate, which, on the addition of water, becomes copper vitriol, easily recognized by its color and by several simple and well-known tests.

As the process continues, and the temperature is gradually raised, this salt also undergoes decomposition, yielding at first a basic sulphate of copper, which, upon losing its acid, becomes a dioxide and eventually a protoxide of that metal. These last changes, however, require a protracted high temperature.

The oxidation of the iron present is pretty well advanced at the time of the maximum formation of cupric sulphate; but it is not until the decomposition of at least 75 per cent. of the last-named salt that the formation of sulphate of silver begins with any considerable energy. When once fairly started, however, this interesting and important reaction progresses with great rapidity, and the decomposition of the comparatively large proportion of sulphate of copper present furnishes ample oxidizing influence for the minute quantities of sulphide of silver. The maximum formation of the latter substance usually coincides with the almost entire destruction of the former salt, and it is at this point that the *Ziervogel* calcination should terminate, as any further exposure of the silver salt to heat lessens its solubility in water, and may even threaten its existence. The complete decomposition of the argentic sulphate is only accomplished by a long exposure to a high temperature, which is now easily borne by most ores and mattes, the easily melted sulphides having been converted into almost infusible oxides and basic sulphates.

Galena (sulphide of lead), when present, is converted almost entirely into a sulphate of that metal, which, by a higher temperature, is partially decomposed with the evolution of sulphurous acid and the final production of a mixture of free oxide of lead with sulphate, the proportions of these two substances varying according to the quantity of foreign sulphides present.

Zinc-blende requires a higher heat for its thorough oxidation than any of the preceding sulphides, but with care may be eventually changed into an oxide, although a certain amount of basic sulphate of zinc nearly always remains. This includes all the sulphides assumed to have been present in the ore under consideration, nor will others be encountered in practice unless under very exceptional circumstances. Sulphide of manganese is an occasional unimportant constituent of mattes, and presents no particular difficulty in calcining, being easily oxidized to a basic sulphate, insoluble in water, which is stable except at the highest roasting temperatures, when it yields up its acid in the shape of SO_2, and remains as a mixture of manganous and manganic oxides.*

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* This reaction of MnS is given in a small pamphlet devoted to the study of the reactions that take place in roasting the Mansfeld copper matte for the extraction of silver by the Ziervogel method; but it is impossible to credit any individual authorities with the statements made in the preceding few paragraphs, they being for the most part matters of general information.

The gangue-rock of copper ores, being usually siliceous, undergoes no change and exerts no influence upon the calcining process, except in so far as it assists in the oxidation of sulphurous to sulphuric acid by *contact*, as already mentioned.

Calc-spar loses its carbonic acid and is converted into gypsum (calcium sulphate), while heavy-spar—sulphate of baryta—undergoes no change, except in the presence of a powerful reducing atmosphere and at a high temperature, when it may be changed into sulphide of barium. This is soluble in water, and it has been suggested to use its solubility to remove it when its presence is particularly objectionable. A number of trials in this direction were made by the author in 1872 on the heavy-spar ores of Mount Lincoln, Colorado, with very poor results; as it was found extremely difficult to reduce the barium sulphate to sulphide without mixing an amount of coal-dust with the ore at least equal to the weight of the heavy-spar present—from 30 to 40 per cent.—while the BaS formed at this high temperature is only partially soluble in water.

Arsenic and antimony, when present, are usually combined with some metallic base, and behave like sulphur to a certain extent; but they give off a much smaller proportion as volatile antimonious and arsenious acids, while they combine to a much greater extent with the metallic bases, forming salts difficult to decompose and extremely injurious to the quality of the copper.

Under such circumstances, the roasting should be continued in the usual manner until all the sulphides present are oxidized and the resulting sulphates for the most part decomposed. At this stage, from 4 to 6 per cent. of charcoal dust or fine bituminous or anthracite coal-screenings should be thrown upon the charge and thoroughly incorporated with it by vigorous stirring, the heat at the same time being raised to the highest practicable limits. The antimonates and arsenates of iron and copper are rapidly reduced by this means, and a considerable proportion of the injurious metalloids is volatilized, much to the benefit of the resulting copper. The charge should remain in the furnace until all the incorporated carbon is consumed.

In the foregoing description, the process of calcination has been carried much farther than is generally needed, or even desired, in an ordinary oxidizing-roasting as a preliminary to fusion.

Sufficient sulphur must always be present in the smelting mixture to prevent the formation of too rich a matte, which entails heavy losses in metal, and other injurious consequences. But it is not a simple matter to determine in advance exactly the amount of sulphur necessary to produce a matte of any given grade. This depends not only upon the character of the furnace process to be employed—that is, whether blast or reverberatory—but also to a considerable extent upon the manner in which the residual sulphur is combined with the bases present; the rapidity of the fusion; the quality of the fuel; the volume and pressure of the blast; the character of the gangue and flux; and numerous other factors. Whatever may be the condition of affairs, however, it may be pretty safely predicted that the percentage of the resulting matte in copper will almost invariably be very considerably lower than is either expected or desired, so that there is little danger that the calcining department of any newly constructed plant will have too great a capacity in proportion to the rest of the establishment, and many serious errors and disappointments can be traced directly to this habit of over-estimating the probable quality of the matte and failing to provide sufficient calcining appliances.

In case of calcination previous to smelting in reverberatories, it is well to avoid an excess of air toward the close of the roasting process—a precaution easily effected by closing the working openings as far as possible, the rabble passing through a hole in the center of a divided door, while the passage of any considerable proportion of undecomposed air through the grate is rendered unlikely by the lively fire that belongs to this period. By these precautions, the oxidation of any large proportion of the iron present to a sesquioxide is prevented, the latter being infusible and unfit to enter the slag until it is reduced to a protoxide. This reduction takes place instantaneously in the powerful carbonic-oxide atmosphere that prevails in the blast-furnace; but in the almost neutral atmosphere of the ordinary reverberatory, the sulphur alone plays the part of a reducing agent, and a charge composed of the sesquioxide of iron will be found materially to delay the process of fusion, besides producing a thick and foul scoria. The natural remedy is the admixture of a few per cent.

of fine coal stirred thoroughly into the mass of the ore, and fired on vigorously.

An examination of the preceding analyses shows what a large proportion of the sulphur in the charge will go into the matte, especially in the case of rapid smelting in blast-furnaces.

Some kind of an idea may be obtained of the probable composition of the matte to be produced at any given time by the ordinary "matte fusion assay," as given in all works on assaying, wherein the ore to be tested is rapidly melted with merely enough borax and siliceous flux—say, 100 per cent. of borax and an equal amount of pulverized window-glass—to flux its earthy constituents, some 10 per cent. of argols or other reducing agents being also added.

But the results are far from satisfactory, and after patiently using it for some two years, and being oftener misled than guided by its results, I discarded it completely, and trusted principally to the eye, occasionally aided by the following calculation, which gives better results than any other familiar to me:

Taking the contents of copper in the charge as a standard for comparison, sufficient sulphur should be allotted to it to form a subsulphide, the excess of sulphur still remaining being supplied with sufficient iron to form a monosulphide of that metal. If other metals are present, such as lead, zinc, or manganese, $\frac{3}{4}$ of the former, $\frac{1}{2}$ of the second, or $\frac{1}{4}$ of the latter substance may be first considered as forming a monosulphide with the sulphur, there being in such a case just so much less of the metalloid left to take up iron. This rule gives quite accurate results in rapid blast-furnace smelting, and where abundance of iron is present. If the rate of smelting be slow, and considerable lime, magnesia, or baryta be present, 5 per cent. of the sulphur contents of the charge should be deducted before beginning the calculation; and if the smelting-furnace is a reverberatory, the resulting matte will average 8 per cent. higher in copper than found by this formula.

A simple illustration will make this method of calculation more clear.

We will assume that a roasted ore having the following composition is to be smelted in a blast-furnace:

ANALYSIS OF CALCINED ORE.

Cu = 9·0 per cent.
Fe = 45·0 per cent.
SiO_2 = 27·0 per cent.
Zn = 2·0 per cent.
Pb = 2·0 per cent.
S* = 7·8 per cent.
O and loss = 7·2 per cent.

Total, 100·00 per cent.

CALCULATION OF MATTE WHICH SHOULD RESULT FROM FUSION OF THE CALCINED ORE.

Following the rule given,

9 Cu require 1·8 S to form a subsulphide.
$\frac{3}{4}$ of 2 Pb " 0·2 S to form a sulphide.
$\frac{1}{2}$ of 2 Zn " 0·3 S to form a sulphide.

This provides for 2·3 per cent. of the 7·8 per cent. of sulphur present, leaving 5·5 per cent., which will take up enough Fe to form a monosulphide. Calculation shows that 9·6 per cent. of Fe will thus be required, leaving 35·4 per cent. available for the slag.

In order to express the composition of the matte just calculated, in the ordinary manner, we multiply the amount of each ingredient by a common factor that will reduce it to a percentage. In this case the factor is 3·46.

9 Cu + 1·8 S = 10·8 × 3·46 = 37·37 per cent. Cu_2S.
1·5 Pb + 0·2 S = 1·7 × 3·46 = 5·88 " " PbS.
1 Zn + 0·3 S = 1·3 × 3·46 = 4·5 " " ZnS.
9·6 Fe + 5·5 S = 15·1 × 3·46 = 52·25 " " FeS.

7·8 per cent. S. 100·00 per cent.

Thus the matte from such a charge will contain about 30 per cent. copper; the slight loss of sulphur by volatilization and as SO_2 being usually fully balanced by the presence in the matte of a certain proportion of subsulphides in place of sulphides, or even of metallic iron.

The same charge smelted in a reverberatory furnace would yield a matte of nearly 40 per cent. Cu.

The proper composition of the slag has not been particularly considered in this example. It would be somewhat too siliceous for

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[*] As most of the oxidized compounds of sulphur contained in the calcined ore will be reduced to sulphides in the cupola furnace, it is proper to estimate all the sulphur present as metallic sulphur.

blast-furnace work, requiring the addition of a little limestone; while for reverberatory work, it would be about right as it stands.

From the foregoing statements, it is evident that in ordinary copper smelting the calcination of sulphide ores need seldom be pushed to the point of perfection indicated when treating of the chemical reactions that take place in the roasting. On the contrary, a due regard for the proper quality of the resulting matte and slag will probably render it advisable to stop the calcining process long before the decomposition of the sulphate of copper in the charge is complete, and even while a considerable portion of undecomposed sulphides still remains. If, however, the calcination has been carried too far, it is very easy to regulate matters by the addition to the smelting mixture of a very small proportion of raw sulphuret ore.

A glance at the behavior of the various compounds of sulphur and bases is essential for the clear understanding of the much greater richness of the matte resulting from the fusion of any given charge in a reverberatory than in a blast-furnace, and of the importance of having a certain proportion of sulphates and other oxidized compounds in the smelting mixture, in order that they may react on each other in the manner best calculated to eliminate the residual sulphur, and thus in a measure make up for imperfect roasting.

In the blast-furnace, but little sulphur can be directly volatilized; and, consequently, simply fuses with the copper or iron present to form the artificial sulphide called matte. But the sulphates in the presence of carbonic oxide may undergo the following reaction: $CO + FeO, SO_3 = CO_2 + SO_2 + FeO$; the carbonic oxide burning to acid, while the sulphuric acid is reduced to sulphurous acid, which escapes by volatilization, and the protoxide of iron unites with silica to form a slag. But this is true of only a very small proportion of the sulphates present, as in the powerful reducing atmosphere of the blast-furnace, the sulphurous acid, even when once formed, comes in contact with an overwhelming proportion of CO, which in burning to CO_2 robs the SO_2 of its oxygen, reducing it to sulphur, in which condition it unites with iron or copper and enters the matte, thus increasing the amount of this product, while it robs the slag of its most valuable constituent. It is interesting to note the striking difference of the reaction in the reverberatory furnace, where the atmosphere may be regarded as neutral; CO, the most powerful reducing agent, being virtually wanting:

$$\begin{aligned} Cu_2S + 4\,CuO,SO_3 &= 6\,CuO + 5\,SO_2. \\ Cu_2S + 2\,CuO,SO_3 &= 2\,Cu_2O + 3\,SO_2. \\ Cu_2S + 2\,Cu_2O &= 6\,Cu + SO_2. \end{aligned}$$

By studying these formulæ—taken from Percy, Kerl, and Rivot—it will no longer seem strange that the reverberatory produces so much richer matte than the blast-furnace from the same charge. Nearly all the reactions between sulphides and sulphates result in the formation of oxides and volatile SO_2, and were it not for an almost invariable preponderance of undecomposed sulphides in the charge, the elimination of the sulphur might theoretically be almost complete. It is by this all-important but frequently neglected establishment of a proper proportion between the sulphides and sulphates, that extraordinary results may be obtained in reverberatory smelting, and the roasting plant greatly reduced.

Although treating of smelting, this matter belongs strictly to the calcining department, and presents a field for study of great interest and practical value. A close analogy may be found in the various reverberatory processes as applied to the smelting of galena ores, where almost exactly the same results are produced, using lead instead of copper, and obtaining metallic lead with a minimum amount of calcination, and putting to accurate practical use the reactions just explained, although text-books on copper metallurgy are strangely silent on this important subject.

The length of time requisite to roast a charge of ore of a given weight in the long furnace under discussion depends, of course, upon the composition of the charge and the degree of thoroughness in oxidation desired. Each of the four hearths of this furnace has an effective area of about 250 square feet, and can consequently receive 4,000 pounds of ore if only 16 pounds to the square foot are charged. This is a very moderate charge, especially for heavy sulphide ores, but will ordinarily give better results than a heavier burden. It will cover the hearth about 2¼ inches deep when charged, increasing in bulk to about 4 inches at the completion of the process. By shifting each charge every four hours, the ore will remain 16 hours in the furnace, a time generally ample to produce the desired effect. On this basis, the furnace would put through 12 tons in twenty-four hours, which may possibly be increased to 16 tons by substituting three-hour drops for the four hours recommended. But this is the extreme limit for two men per shift, nor will these figures be reached under ordinary circumstances. Two cords of wood or 2,240 pounds of soft coal should supply the grate for twenty-four hours, the supply of air to the ash-pit being kept at the lowest possible point. The sulphur contents of the ore furnish a much greater proportion of the heat than does the fuel on the grate.

The manipulations pertaining to the ordinary calcination of ore are too simple and generally known to be worthy of a place in a condensed treatise.

The following experiments form part of a series extending over some ten years, which it was at one time hoped to amplify and carry out into something of positive value. But increasing professional cares, and the impossibility of having the numerous analyses made that constitute an essential part of the work, have prevented the fulfillment of this hope. The material collected will, however, be used wherever it may prove of value in the course of this treatise. The author desires to acknowledge the assistance of Messrs. J. F. Talbot and F. Ames, and others, in the chemical portion of the work.

No. of sample	Copper.	Sulphur.	Dry weight of charge.	Weight lost.	Copper in roasted ore.				Sulphur in roasted ore.	Hours in furnace.	REMARKS.
					As oxide.	As sulphate.	As sulphide.	Total.			
	Pr ct.	Pr ct.	Lbs.	Pr ct.							
1...	7·6	37·0	4,130	14·5	3·65	3·25	1·65	8·55	6·41	16	Heavy pyritous ore.
2...	7·6	39·0	4,130	11·3	2·27	3·10	2·80	8·17	11·30	12	Same ore.
3...	16·4	31·0	3,925	6·4	7·10	3·44	6·80	17·31	8·20	18	Purple ore with much pyrites and some zinc-blende.
4...	16·4	31·0	3,940	9·5	12·80	2·80	2·10	17·70	4·60	24	Same ore.
5...	38·8	24·3	3,600	6·2	29·20	4·40	3·70	37·30	......	18	Matte from cupola.
6...	62·2	22·0	3,580	3·7	54·90	3·80	6·60	64·30	......	18	Blue metal from reverberatory.
7...	74·8	21·4	3,800	2·4	61·60	5·40	7·90	74·90	......	18	White metal from reverberatory.

The loss of weight from the removal of the sulphur is partially balanced by the oxygen combining with the metallic bases, and is exceedingly variable, as may be seen by this table.

The loss in copper during calcination is very small, and almost entirely mechanical, being for the most part recoverable where proper arrangements are made for the deposition of the flue-dust. Average results from personal experience show a loss of about 1¼ per cent. of the original copper contents of the ore during calcination.

This flue-dust is usually of very much lower grade than the ore from which it results, being diluted with the dust from the fluxes, fuel, etc., and generally contains from 20 to 30 per cent. of its value in a soluble form, thus prohibiting the use of water as an aid to its condensation, unless provision is made to precipitate the dissolved metal.

Unless the ores treated are of remarkable purity, it is best to smelt the flue-dust by itself, making it into balls with clay or lime and adding the necessary fluxes. Otherwise, the quality of the metal is likely to suffer, as the substances most injurious to it—arsenic, antimony, and tellurium—are volatile, and sure to be condensed in the flues, thus being collected in a concentrated form.

COST OF CALCINING.

The running expenses of a calciner, aside from the slight repairs just alluded to, are small and regular. In twenty-four hours, it will burn one ton of soft coal (2,240 pounds) at $5, or 2 cords of pine wood, and require the services of four men at $2, and one-quarter the time of a laborer to weigh and bring the charges to the hoppers, the furnace-men dumping and drawing their own charges. This amounts to—

Coal, 1 ton	$5.00
4 furnace-men at $2	8.00
¼ weigher at $2	.50
Wear and repairs on tools, car, etc	.30
Oil, lights, and miscellaneous	.80
Proportion of superintendence (say one foreman to eight furnaces)	.50
Total	$15.10

Considering that twelve tons per day of highly sulphureted ores can be quite thoroughly calcined in such an apparatus, it shows a cost of about $1.25 per ton of ore, which leaves but little opportunity for the inventors of automatic roasting-furnaces to cheapen the results that can be obtained in the old-fashioned calciner, when built of proper dimensions and provided with a powerful draught. The above figures have been repeatedly obtained by the author (reduced, of course, to current prices), and after a tolerably extended metallurgical experience and a trial of almost every reasonable type of roasting apparatus, he still emphatically recommends the simple and well-known open-hearth reverberatory calciner for the preliminary roasting of copper ores and mattes.*

The consumption of fuel depends largely upon the fireman. It is as easy for him to burn two tons of coal as one; but in a properly constructed furnace, with a moderately favorable ore carrying 20 per cent. or more of sulphur, the quantity above indicated will suffice perfectly.

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* See latest results of roasting in Brückner's cylinders.

CHAPTER IX.

THE SMELTING OF COPPER.

By this term, we understand the fusion of the copper-bearing material and of whatever fluxes may be necessary, when the copper, owing to its higher specific gravity, separates from the slag, and is recovered by appropriate means. In the case of oxidized ores, it is obtained at once in a metallic condition, somewhat adulterated with sulphur, iron, and other foreign substances, but requiring only a single operation, or, at the outside, two more operations, to bring it into merchantable form.

But when it occurs in combination with sulphur or arsenic, and accompanied with an excess of foreign sulphides, the result of the first fusion is merely a concentrated ore, freed from the earthy gangue, and resulting from a combination of the copper with sufficient of the sulphur present to form a subsulphide, to which is added as much monosulphide of iron as corresponds to the remaining sulphur, always excepting such portion of that metalloid as is volatilized during the process of fusion. If tin, zinc, lead, silver, antimony, or arsenic are present, they combine with the sulphur for the most part and enter the matte, their affinity to sulphur being in the order mentioned, according to Fournet's experiments.

These various sulphides unite either physically or chemically to form the substance technically known as matte, or metal, or regulus, the latter term not to be confounded with the term regule, which belongs to a matte of a certain richness in copper, and possessing peculiar and well-marked characteristics.

From the above statements, it is plain that, other things being equal, the grade of the matte depends on the amount of sulphur in the ore.

It might, at first glance, seem more economical to push the roasting process to the extent of removing all the sulphur, thus bringing about the same conditions that prevail in the smelting of an oxidized ore; but practice has shown the futility of such a scheme, as, aside from the great expense and difficulty of effecting such a complete calcination, the resulting slags are always too rich in copper; and the copper when produced cannot compare in quality with the metal resulting from the ordinary methods of treatment, where the numerous alternate oxidizing and reducing influences remove, for the greater part, those traces of impurities that are almost invariably present, even in the purest ores, and which have such a powerful effect on the physical condition of the finished metal.

Copper smelting, therefore, is naturally separated into two great divisions, according to the composition of the material to be treated: 1. Smelting of ores containing sulphur (arsenic, antimony). 2. Smelting of ores free from sulphur (etc.). But each of these classes may be again divided, according to the apparatus employed, into—

A. Smelting in blast-furnaces.

B. Smelting in reverberatory furnaces.

But few exact statements have been published by practical metallurgists of comparative results obtained by running the two classes of furnaces side by side on the same ore, and under the same management and conditions. The fact that, during the author's career as manager of various copper works, he has smelted about an equal amount of ore in each class of furnace, and in several instances carried out quite extensive comparative tests at the same works as to cost, capacity, etc., may lend value to such statements.

Considerable animosity has been evinced by the partisans of the reverberatory and of the blast-furnace system of treatment—or Swansea and German methods, as they are often termed.

Much of this arises from a want of exact knowledge and appreciation of the advantages and peculiarities of the opposing systems.

Since blast-furnace smelting has obtained a footing in the United States, it has become so changed from its original as to be scarcely recognizable, and as here used, by the more advanced metallurgists, can challenge competition with the reverberatory under most circumstances, and, where the conditions are at all favorable, can show results far surpassing the best Swansea work in yield, economy, and capacity.

That this may seem novel or even doubtful to English smelters, is quite natural, when it is recollected that the full extent of these remarkable advances is known to comparatively few metallurgists, and that very little relating to the same has been published.

It is with the modern American form of the German copper process that all comparisons must be instituted; and this comprises not only a great improvement in the processes of calcination and the construction and management of the blast-furnaces used, but, in many cases, the employment of reverberatories for certain portions of the matte concentration, while the process of refining is in nearly all cases carried on according to the Swansea method.

In any attempt at a comparison of these two great methods of smelting, one is confronted by the inextricable mingling of the commercial with the metallurgical that is so characteristic of the English system. Without a thorough understanding of the peculiar local conditions under which the ores are purchased at the Swansea ticketings, it is impossible to fully appreciate the fine points of the complex and ingenious system that time and circumstances have elaborated, or to realize the important influence exercised on the whole subsequent series of operations by the amount of judgment displayed in the purchase of the ores, and in the adaptation of the same to the immediate needs of the works.* The Swansea smelter receives his ore in numberless small parcels, differing not only in richness, but in purity and other qualities. To carry out the reverberatory process to the best advantage, he requires, in addition to the main supply of sulphide ores, a certain proportion of oxides and carbonates, all of which are obtainable in the public ore market. His coal is of the cheapest and most suitable quality, and the refractory material—fire-brick, clay, siliceous sand, etc.—is obtainable at prices far below American rates. He also has at his command a body of experienced and skillful workmen who have grown up at the furnaces, and who, at very low wages, are fully capable of executing all the difficult operations demanded by this system of treatment. In addition, he has a market for his product, where every variety of metal brings the highest justifiable price.

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* See Percy on Copper, for a full description of the Swansea ore sales, together with quality and value of ore offered.

It is very evident that such a state of affairs cannot be compared with average American conditions, where, in the greater number of instances, the ore supply comes from only one or two sources, constant in its composition, and usually in very large quantities. This, with the high wages and exceedingly expensive fuel, has caused the introduction of labor-saving machinery and appliances to an unprecedented extent, as well as a constant endeavor to lessen the proportion of fuel to ore smelted. The lack of steady and skilled furnace-men, and the high cost of refractory materials, have also had a powerful influence in shaping the processes of treatment, and have perfected the water-jacketed cupola, without which many of our most successful metallurgical enterprises could hardly exist. The same influences have concentrated the works for the refining of copper in a very few hands, and located them with the view to cheap coal and refractory materials and to a market for the finished product. Another factor that has had its effect in greatly simplifying our domestic process of refining is the extreme purity of the lake copper, which, in this country, takes the place of the higher grades of English copper, there produced by special refining processes, and commands correspondingly higher prices.

BLAST-FURNACE SMELTING.

a. Of sulphide ores.
b. Of ores free from sulphur.

a.—TREATMENT OF SULPHIDE ORES.

The fusion of sulphide ores in blast-furnaces may take place either with or without a previous calcination, as has been already referred to.

Where the percentage of sulphur is small in proportion to the copper contents, a sufficiently high-grade matte may be obtained by the direct fusion of the raw ore, with the addition, of course, of the proper quantity of basic substances, such as iron ore, limestone, etc., to flux the very large proportion of gangue rock, which, in most cases, consists of quartz or some highly siliceous substance. As the amount of basic material required to flux silica is very large, about two pounds to one of silica, highly siliceous ores can be remuneratively smelted only under exceptionally favorable circumstances. Otherwise, such ores would often be more advantageously treated by one of the wet processes. No better flux for silica can be had than the ferruginous slag arising from the concentration-smelting of copper mattes, which usually contains about one per cent. of copper, but can seldom be obtained in such quantities as to form a permanent flux for any considerable amount of highly siliceous ore.

The class of copper ore most commonly subjected to blast-furnace treatment in the Eastern portion of this country is a highly pyritous material, usually having from 2 to 6 per cent. of copper, and varying amounts of silica. This is first burned for the manufacture of sulphuric acid, after which the cinders are smelted for copper. The ores from Capelton, Province of Quebec; Milan, New Hampshire; Virginia; and Georgia carry an excess of iron, and to them may be added the monosulphide ores found at Ely and Copperas Hill, Vermont; Ducktown, Tenn.; and Ore Knob, North Carolina. In the West, a large number of mines furnish copper ores usually of somewhat greater richness, but in which the silica is in excess, rendering the smelting more difficult and occasionally making the employment of reverberatory furnaces advisable. To this class belong, also, the ores of the Douglas and many other Maine deposits; the St. Genevieve, Mo., mines; a large class of argentiferous copper mines in San Juan District, Colorado; most of the Butte City veins; and a series of important though little known deposits in Lower California and Nevada. This brief enumeration includes most of the types of sulphide ore likely to come to the blast-furnace; and the first object of the metallurgist is to see how he can form a proper slag at the least possible cost.

A proper slag for a blast-furnace should contain between 24 and 36 per cent. of silica, although, under pressure of circumstances, these extreme figures may be either raised or lowered about 6 per cent. without seriously compromising the running of the furnace. But every per cent. of silica in excess of 36 will be felt in a rapid reduction of the amount smelted in twenty-four hours.

As it is usually a long time before the young metallurgist fully appreciates the enormous damage that even a slight excess of silica will effect, the writer desires particularly to emphasize this point, and to declare that, according to his own experience, three-fourths of the troubles and annoyances experienced by the blast-furnace manager result from this cause. There are many instances of furnaces that have given trouble from the day of their first starting, being relieved by a slight addition of iron ore, and smelting operations have changed from a loss to a profit, capacity being increased 40 per cent., and the campaign lengthened from 20 days to several months by slightly increasing the insufficient charge of limestone and iron.

In speaking of modern blast-furnace smelting, we may well omit any lengthy description of the small brick furnaces so familiar to all who look over the illustrations in Kerl and Plattner. The economy of larger furnaces has been thoroughly demonstrated, and in the present treatise, plans and descriptions will be mostly confined to the two principal types of furnace now in use:

1. The water-jacket furnace, with its various modifications.
2. The long rectangular brick furnace.

With a thorough understanding of the construction and management of these two varieties of furnace, the metallurgist is amply prepared to obtain the best results known to modern engineers.

1. The water-jacket furnace, with its various modifications.

Without attempting to determine to whom the credit belongs of adapting the principle of water-cooling to copper blast-furnaces, it may be hailed as the greatest advance in the treatment of that metal that has been made since the introduction of the English method of refining on the hearth of a reverberatory furnace. With its employment, the burning out, and consequent "freezing up," of the furnace from the half-fused masses of molten fire-brick, have become things of the past, and campaigns have been extended to an unprecedented length. In fact, where no accident occurs, nothing compels the stoppage of the furnace excepting the need of general repairs to machinery, etc.; the cleansing of the interior of the jacket from sediment; and the possible choking up of the furnace shaft with accretions of sulphides of zinc or lead, which occur in minute proportions in almost all copper ores.

The material of which the jacket is composed may consist of cast-iron, wrought-iron, or mild steel. The brand of wrought-iron known as fire-box iron is preferred by the author, as less liable to scale and blister by the heat, and because capable of being bent without weakening. Where cast-iron is used, the furnace is composed of several sections, held together by clamps or rings; but aside from the excessive weight, this material is somewhat liable to crack when exposed to extreme fluctuations of temperature, although, as of late the castings are made from five-eighths to three-quarters of an inch thick, with a water-space of from four to ten inches, this accident is much less likely to occur. The thickness of the wrought jackets need not exceed that of ordinary boiler plate, and this material is peculiarly suited to circular furnaces, the inner plate having been found to buckle and weaken, owing to difference of expansion, when used in long rectangular furnaces—an observation made by Mr. J. B. S. Herreshoff, of New York, and which he has obviated by using a very elongated oval shape in place of the rectangular.

Although the circular form possesses certain advantages for smelting-furnaces, experience has taught us that the ordinary blast used in copper smelting, which seldom exceeds three-quarters of a pound per square inch, cannot well penetrate to the center of a charge in a furnace of greater diameter than fifty inches, this being the outside limit in cases where at least one-half the charge is in lump form. In wrought-iron jackets, the width of the water-space has been diminished little by little, until even two inches has become a not uncommon standard, and its reduction over several square feet of surface to one and one-quarter inches has not been accompanied with any evil results. The cold feed-water is generally introduced near the middle or lower portion of the jacket, and doubtless settles to the lower point at once, rising gradually as it becomes heated, and escaping through a pipe of somewhat greater area from the upper portion of the jacket. It is best to have the escape-pipe tapped into the water-space in such a way that it is even with the extreme upper surface, thus preventing the accumulation of any steam that might form. Circulating pipes are introduced into the water-space by some of the best manufacturers; but while not prepared to deny their value, the author has run water-jacketed furnaces of many sizes and shapes, and under varying conditions, and has never felt the need of any guide-plates, the difference in temperature of the incoming and outgoing water always being sufficient to keep up a lively circulation to the most distant point, while any sediment introduced in the water could always be easily removed through the hand-holes provided for that purpose.

The following figures, deduced from personal experience, give furnaces of this description, and of various diameters, and the quantity of water required when in full blast:

Diameter. Inches.	Water per hour while blowing in and out. Galls.	Water per hour during normal running. Galls.
24	900	460
30	1,200	600
36	1,450	950
42	2,200	1,200
48	3,000	1,500

These figures refer to a supply of fresh water; but where the same water is used over and over again, about 3,000 gallons per twenty-four hours are required to make up the loss by evaporation, etc., in a 36-inch furnace in the dry, hot climate of Arizona. Prof. F. L. Bartlett has arranged a tank at such a height that its upper surface is a trifle higher than the water level in the jacket, by which means a constant circulation takes place, requiring only the addition of sufficient fresh water to replace the evaporation. This may require a little extraneous aid from a force-pump or steam-jet during the period of blowing in; but in a few hours, when the upper portion of the furnace is cooled down to its normal condition, this arrangement is said to answer every purpose, though the water may from time to time become a little hot, and even form a certain amount of steam. To save all calculation, it may be stated that a 2½-inch feed-pipe, with a 2¾-inch discharge-pipe, the former coming from a tank that will give eight or ten feet pressure, will give all the water necessary for a 42-inch furnace.

Where the hot discharge-water is not used over again, it is economical to employ it for the boiler, or in winter to lead it into some lower tank, or where it may be used for ore-concentration purposes, if such a plant is present. The furnace-jacket should always be provided with a drain-cock to empty it when not in blast in cold weather.

While the weight and clumsiness of cast jackets prevent their being made of any great size, so that the first jackets only occupied a narrow circular ring at the level of the tuyeres and for a few inches above, it has now become quite customary to cast them in sections of from 30 to 60 inches in height, while the circular or oval wrought jackets usually extend from a point some 10 inches below the tuyeres, to the threshold of the charging-door, a distance of from 6 to 10 feet. This saves all brick-work, excepting the small amount in the bottom, and the flue on top, by which the gases are conducted to the stack. This flue and upper brick-work are usually supported on light iron columns, the jacket itself being either suspended from the same columns by a ring, or resting on cast legs of its own.

The bottom of the furnace may be constructed in various ways; but in the smelting of roasted pyritic sulphide ores, American practice is pretty unanimous in entirely doing away with the ordinary deep crucible, substituting for it merely a sloping bottom a foot or less below the tuyeres, from which the entire molten material escapes through a narrow groove under the breast, then first entering an outside crucible or "well," in which the matte separates from the slag, and is tapped into molds, while the slag flows from a spout into iron pots arranged on wheels for convenient dumping. It is this transfer of the crucible from the inside to the outside of the furnace that has divested cupola work of most of its terrors. By this simple means, we escape the troublesome chilling over of the metal in the crucible, and the frequent freezing up of the tap-hole, rendering it impossible to empty the furnace without the most laborious and tedious work. The formation of sows and other kindred products is also prevented by the immediate escape of the fused ore from the powerful reducing action of the fuel, as are also the cutting down of the crucible and thinning of its surrounding walls until the metal and slag burst through, and a long list of lesser troubles, familiar to every practical furnace-man.

The advantages gained by modern blast-furnace practice may be partially estimated by comparing the following statement with the results given in succeeding pages of this treatise.

As a matter of historical interest, it may be put on record that the first "well" used in connection with a copper furnace in this country was built by James Douglas, Jr., at his Phœnixville works, in 1879. The author is unable to find any authentic information of any earlier use of the modern form of well, or independent fore-hearth.

In a valuable and interesting lecture delivered by Mr. Henry Hussey Vivian, M.P., at Swansea, December 20th, 1880, on the history and processes of copper smelting,* after admitting that the blast-furnace invariably excels all other apparatus in the production of a clean slag (that is, free from metal), he adds : "It has a constant tendency to reduce the oxide of iron contained in the calcined ore into metallic iron, and thus to produce a mass of infusible material at the bottom of the furnace, which, in no long period, causes the entire or partial destruction of the furnace. Even in the best managed continental works, I have proofs that the so-called iron 'sows' are produced; in fact, they are an almost unavoidable incident of melting calcined copper ores in blast-furnaces." And in referring to his personal examination of the ancient slag-piles surrounding the famous Rio Tinto and Tharsis mines, in Spain, he says: "I examined critically the slag-heaps, and was astonished at the freedom of the slags, made, perhaps, two thousand years ago, from prills. At this moment, with all my accumulated experience of copper smelting, I don't know how they made those heavy irony slags so clean."

If Mr. Vivian had had the use of the blast-furnace forced upon him under these conditions, there is little doubt that he would have solved this problem that now perplexes him. At any rate, the Americans have solved it in the most satisfactory manner, and can refer the inquirer to the records and practice of almost any of the principal copper-works in this country, where slags of the most highly ferruginous character, as well as the most siliceous, are produced in blast-furnaces, not only free from prills, and without the slightest accompaniment of iron sows, but also far lower in chemically combined copper than can possibly be made in reverberatories. How such results are obtained, will be explained when treating of furnace management, although the improved construction has also an important influence in effecting these results.

In the treatment of sulphide ores, the practice, formerly common, of having the crucible wholly or in part under the main body of the furnace—as in the German Tiegel-Ofen and Sumpf-Ofen—can under no conditions be recommended. Those interested in this system of furnace construction will find full details regarding the same in any of the standard German works on the subject, and in the section of this treatise devoted to brick furnaces.

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* *Copper Smelting: its History and Processes.* By Henry Hussey Vivian, M.P. A Lecture delivered at Swansea, in the Theatre of the Royal Institution of South Wales, December 20th, 1880. To which is added: A History of the Baltimore Copper Works at Canton, Maryland; Sketches of the Forest Copper Works, and the Hafod Copper Works, Swansea, South Wales. With illustrations. New York: The Scientific Publishing Company, 27 Park Place. 1881. 8vo, pamphlet.

The bottom of the furnace, according to modern practice, is brought up to within from 6 to 12 inches of the tuyere level, in most cases sloping slightly toward the breast, so that the entire molten contents may flow out through a narrow channel under the latter, and discharge into an anterior compartment, consisting either of a deep basin formed of "steep" (*Gestübbe, Brasque*), or a large rectangular box, made of fire-brick held together by iron plates, and in which the separation of matte from slag takes place quietly and thoroughly.

Provision is made to prevent any escape of blast under the breast, either by so thoroughly covering over the orifice and channel that only a minute groove exists, which is constantly filled to its utmost capacity with the molten ore, which soon forms an impervious cover to its channel; or by so raising the terminal slag-spout, and lowering the anterior wall of the furnace, that the blast is securely "trapped," just as sewer gas is prevented from escaping in an ordinary drain.

The first method, combined with the steep crucible, is best adapted to the production of pig-copper or very rich metal, as in matte concentration, owing to the great tendency to chill of these substances; while the latter plan is far preferable for ordinary ore-smelting, where matte of much lower grade is produced in considerable quantities. Where pig-copper is produced on a large scale, and in furnaces of considerable capacity, it is best to drop the "steep" crucible entirely, as the large volume of hot metal will permit the use of the much preferable brick "well" without chilling. In either case, the furnace should be taken in hand by the head smelter as soon as the water-jacketed shell is properly suspended in position with its upper brick-work complete, and the connection established between the same and the stack that is to convey away its gases.

If the furnace is to be used for matte concentration or for the production of pig-copper from roasted matte, and consequently provided with a steep crucible, four iron plates should be provided, forming a rectangular box about 3½ feet high, some 4 inches wider than the furnace on each side, and extending from the back to 36 inches in front of the breast. The front plate is provided with the usual cast-iron slag-spout, fitted with a groove to slip on without bolts, while one side plate should be perforated with a small hole or, better, slit, two inches wide and seven inches high. As this is the tap-hole, its lowest point should correspond with the apex of the inverted cone-shaped crucible.*

A heavy matte-spout, eight inches long, and at least two inches thick on the bottom, and cast as part of a strong square plate, should be bolted on at the lower edge of the opening, an arrangement found necessary for the proper and efficient plugging of the tap-hole. A foundation for this iron box, which is held together by bolts through ears or other simple means, should be made by either laying down thick iron plates of the proper dimensions, or by putting in a tight floor of fire-brick on end, laid in thin clay mortar with an addition of 10 per cent. of silicate of soda. This will prevent the gradual penetration of matte into the rock foundation, and is a necessary precaution. Upon this the portion of the rectangle corresponding to the base of the furnace is built up as a solid column of fire-brick to within 10 or 12 inches of the tuyeres.

In some furnaces, the water-jacket is continued for a considerable distance below the tuyeres. In any case it is proper to continue the brick-work, on the outside at least, to the edge of the jacket, surrounding the lower portion of the latter with a few courses, to insure a tight joint.

In the anterior half of the iron fore-hearth, a single row—4½ inches—of fire-brick is sufficient on the sides, while a 9-inch wall is usually built in front, an elongated opening corresponding to the tap-hole being left in the brick-work.

It is now evident that the fore-hearth is filled in its posterior half with a solid mass of brick-work, extending up to, and becoming continuous with, the circular shell of the furnace proper, while the anterior half contains a large square opening, surrounded by a brick wall, extending down nearly to the foundation of the structure; communicating by a channel under the breast with the interior of the furnace; opening into the tap-hole on one side, and into the slag-spout in its upper anterior wall.

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* The cuts accompanying the description of the Herreshoff and Orford furnaces may be referred to in connection with this and succeeding paragraphs.

The interior of the furnace is now provided with a bottom, consisting of four parts ground calcined quartz to one part plastic fire-clay. This is firmly tamped upon the brick base and carried in a wedge-shaped wall around the sides of the circular interior, resting against the water-jacket, and thinning out to nothing just below the tuyere openings. The component parts of this mixture should be ground through a 16-mesh screen, or finer, and after a thorough mixing, should be slightly moistened, and tamped firmly into place with iron bars, shaped on the end like a four-leafed clover, and slightly heated, to prevent adhesion of the material.

A layer of three or four inches is quite sufficient for an ordinary furnace bottom, it being only necessary to protect the brick-work until there is deposited from the smelting charge an exceedingly refractory mixture of metallic iron, matte, and slag, which forms a massive and permanent bottom, far superior to any artificial substance. It is only in smelting very poor pyritous ores, producing a large quantity of low-grade matte, that any "cutting down" of the bottom occurs. The measures appropriate to this condition of things will be discussed when treating of the large brick furnaces, in which this class of material is commonly produced.

The furnace bottom should slope slightly toward the breast, at which point it meets the "steep" (German, *Gestübbe;* French, *Brasque*) with which the anterior compartment is filled, and in which the crucible is formed, its deepest point communicating with the tap-hole in the side plate.

This moderate and economical use of steep must not be confounded with the old-fashioned practice of establishing an enormous fore-hearth, filled with this material, and requiring constant repairs and attention. Although abolished in many modern works, it possesses peculiar qualities, which render it very valuable in certain blast-furnace operations—such as the smelting of calcined matte—where the product is either pig-copper or a very high-grade matte, and the capacity of the furnace not large.

Both of these substances have a strong tendency to chill, especially when using the exterior crucible, which is for the most part prevented by the use of steep, which, besides being an excellent non-conductor, seems actually to generate heat—possibly from the slow combustion of its carbon—thus preserving the metal fluid, while any chill that may form in the crucible is easily removed without damaging its walls and interior, as would be the case with clay or brick-work.

The permanency of the basin and tap-hole depends greatly upon the quality of the steep, which should be made as follows: Crush the constituents separately through a 20-mesh screen, or as much finer as is practicable. A Bogardus or Sturtevant mill will be found useful for this purpose, and has a much greater capacity than the light stamps often used. Mix very thoroughly while dry, and moisten with water through a rose nozzle to such a degree that the mass will ball when pressed vigorously in the hand, without imparting any dampness to the skin. Tamp firmly with inch square bars, and avoid stratification by adding a shovelful at frequent intervals, and before a hard surface is produced by the pounding. The following proportions are suitable for varying conditions. When the product is metallic copper, use by measure: 3 parts coke, 2 parts raw fire-clay; or 4 parts coke, 2 parts raw clay, 1 part burnt clay or ground brick; or 3 parts charcoal-dust, 2 parts raw clay, 1 part ground red brick. For a product of rich matte, use: 7 parts coke, 5 parts raw clay; or 3 parts charcoal, 2 parts raw clay, 1 part burnt red brick.

A large proportion of carbon counteracts the chilling of the metal and the consequent formation of skulls in the fore-hearth, but is less able to stand mechanical violence than the heavier steep, which has more plasticity. Charcoal-dust makes a somewhat fragile mixture, but an excellent one for retaining heat.

The arrangement just described is particularly suited to a small slow-running furnace, where it is intended to make a rich product, and where reasons exist for producing a slag sufficiently free from copper to be at once rejected. That this is perfectly practicable is demonstrated at various establishments in this country, where, by a somewhat lavish expenditure of fuel, a light blast, and a very slow run, a slag containing below 0·7 per cent. of copper and exceedingly ferruginous is produced in conjunction with pig-copper. The material smelted is stall-roasted matte, with a very small addition of old brick and furnace ends, and in spite of the character of the charge and the powerful reducing action due to the slow run, the formation of all metallic iron is avoided—a result almost impossible to obtain in furnaces with an interior crucible.*

To prevent the delay arising from the frequent though slight repairs indispensable from this form of furnace, it is sometimes customary to widen the fore-hearth sufficiently to contain two crucibles side by side and used alternately.

The copper may be removed from the crucible either by tapping into molds of sand or iron, or by ladling, the latter method being more frequently employed where the product is pig-copper, owing to the difficulty of opening the tap-hole after a run of some length. For ordinary ore-smelting, producing a matte below 50 per cent. copper—usually between 33 and 40 per cent.—no arrangement can approach the modern fore-hearth for convenience, economy, and safety; nor can the solid brick base just described compare with the simple iron drop bottom, as used in cupolas devoted to the melting of pig-iron or castings. A most useful modification of this fore-hearth is shown in the illustrations of the Herreshoff furnace. The profession is indebted to Mr. J. B. F. Herreshoff for this as well as for several other improvements in connection with this furnace. The author also desires to express his obligations to the same gentleman for many valuable practical suggestions that he will not attempt to specify in detail. The fore-hearth or "well" is here placed on wheels, for convenience of removal, though more frequently it rests upon the solid ground.

Another feature of especial value is the arrangement of the bottom of this furnace, which consists merely of a circular, concave cast-iron plate, firmly bolted to the lower border of the water-jacket, which extends some twelve inches below the tuyeres. This bottom is covered with a single course of fire-brick, resting on a shallow layer of sand, and might seem to be but a feeble barrier to such material as molten ore. It would, in fact, last but a very short time, were it not that the outlet of the furnace, through which all its liquid contents must pass, consists of a

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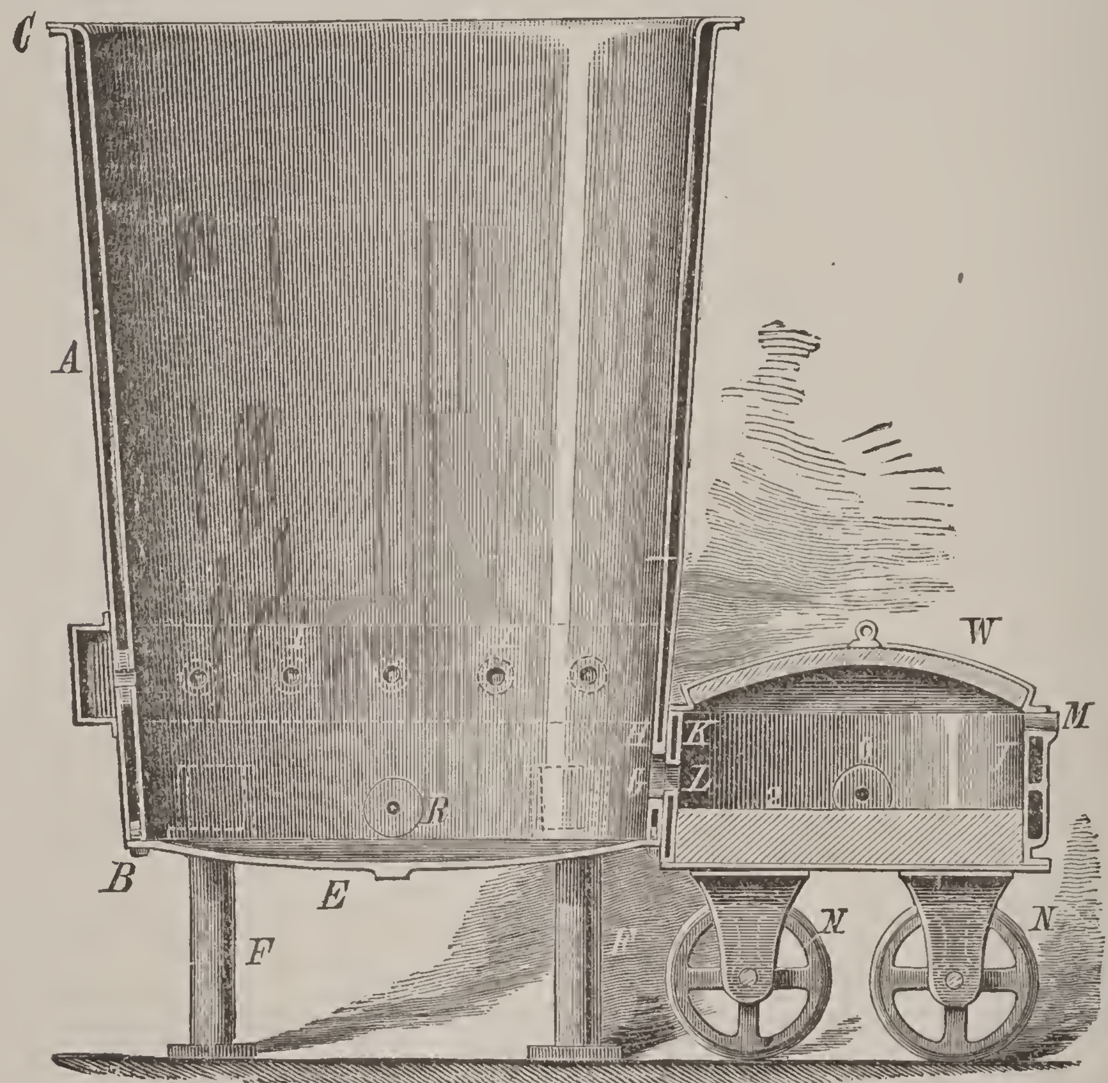
* The best example of this interesting but somewhat antiquated practice, though executed in brick furnaces, is found at Ely, Vermont, where there are some eight furnaces for the production of pig-copper in the manner indicated.

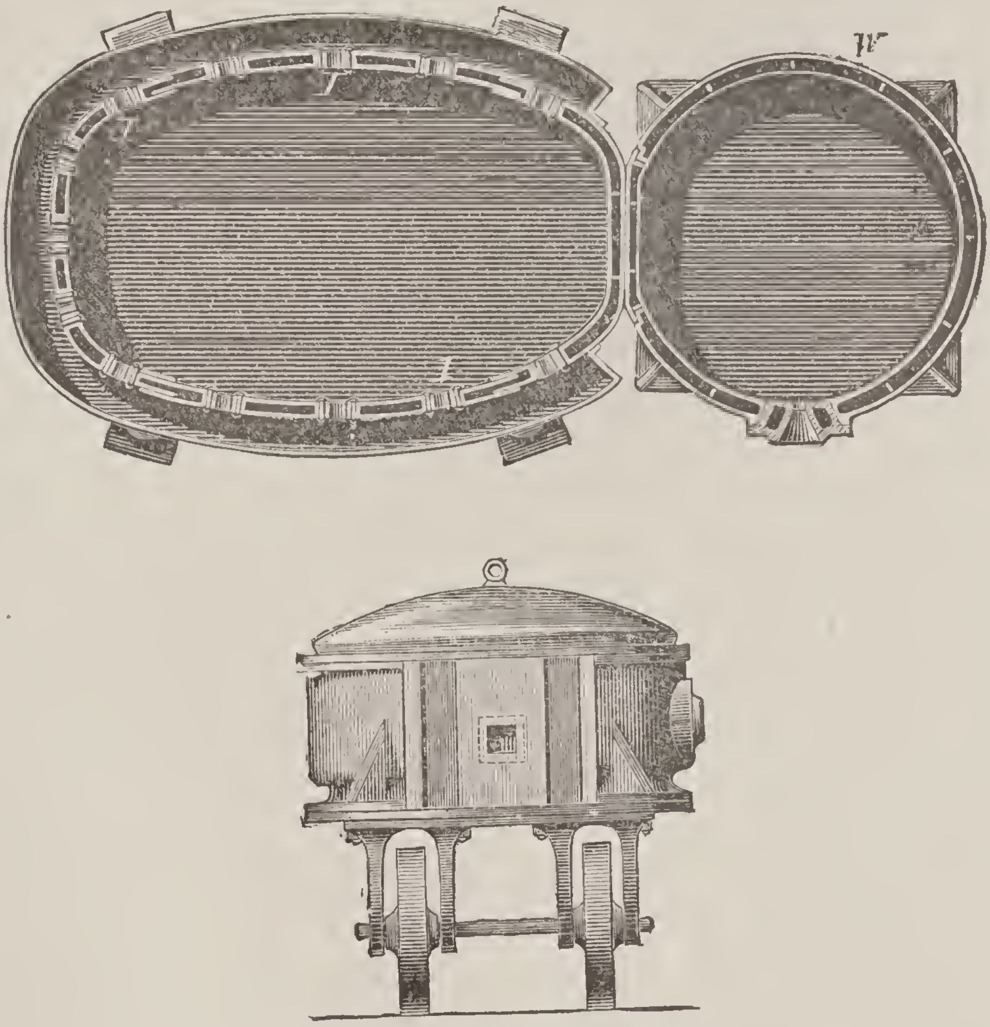
4-inch by 7-inch circular opening through one side of the water-jacket, and is consequently so protected that the slag and matte can cut their way no deeper than the lower rim of the opening. There stands, therefore, constantly within the furnace a pool of molten material at least as deep as the lower border of the orifice referred to, while the constant loss of heat therefrom by radiation through the thin bottom of the furnace speedily converts it into a solid and permanent block, which need only be removed when cause exists for detaching the bottom. The most novel feature of this arrangement consists in a similar opening in the back wall of the movable fore-hearth, which, being also protected by a small separate water-jacket plate, and backed up until it exactly meets the furnace opening, forms a continuous, though very short, water-cooled channel from furnace to fore-hearth. The slag discharge of the latter is several inches higher than this channel, so that when the well is full and slag begins to run over into the pots, the opening just described is covered several inches deep with liquid material, which stands at the same depth in the



HERRESHOFF NEW FURNACE.

interior of the furnace as in the fore-hearth, except in so far as lowered by the pressure of the blast. The wind is thus completely trapped, and its constant blowing through, which is one of the most common and obstinate annoyances of blast-furnace practice, is effectually prevented.

The products of the fusion, usually only two in number in copper smelting, separate in this large fore-hearth very completely, the matte settling quietly to the bottom, while the slag flows through the anterior spout in a constant stream. When globules of matte begin to appear in the slag stream, as evinced by the sparkling of the same while falling into the pot, and its greater liquidity when a small portion of the suspected slag is caught in a shovel and inclined from side to side while cooling, the tap-hole in the side plate is opened with a pointed steel bar, driven in with a heavy hammer if necessary, and the metal allowed to flow into molds of sand or, in some cases, of cast-iron.



HERRESHOFF NEW FURNACE.

When the well is thus empty and the communicating channel between furnace and fore-hearth uncovered, the blast escapes through the same with full force, chilling the surface of the slag in its passage, and hurling glowing fragments of coke and globules of molten ore in every direction. This is completely obviated in the Herreshoff system by plugging the slag-spout opening with a ball of plastic clay heavily weighted. The fore-hearth being tightly covered with slabs formed of fire-brick held together by iron clamps, the blast is in this way entirely confined to the interior of the furnace, while the fore-hearth soon fills, and the wind is trapped as before.

Still another convenient feature is shown in the arrangement by which the matte, when tapped, is kept free from the aftercoming slag, of which a considerable quantity is present in the interior of the furnace and well, even after the appearance of matte at the slag-spout. As it is sometimes impossible or unadvisable to close the tap-hole at the exact moment when the last of the matte has escaped and the first of the slag begins to flow, Mr. Herreshoff has arranged a tilting iron launder between the matte-spout and molds, which, when held up by a chain passing over a pulley, conducts the liquid to the regular molds, but when released by a catch, turns upon a horizontal pivot, and conveys the slag in the opposite direction and into compartments in the sand, where it is obtained in proper shape for re-smelting..

Brick fore-hearths of various patterns, but in the main resembling the type just described, have been in use in smelting sulphide copper ores for some seven or eight years, and are certainly superseding all other arrangements. A brick fore-hearth of this description, strengthened by iron plates cast dishing to prevent cracking, and firmly bolted together through projections at the corners, will last, according to the quality of the products and the rapidity of the process, for from two to thirty days, a week being perhaps the average life. Their destruction is brought about in two ways: by gradual chilling about the sides and bottom until the cavity becomes too small or tapping is rendered impossible; or by the cutting away of the brick lining from the action of a hot basic slag and a low-grade ferruginous matte. The former condition results usually from the presence of a siliceous, infusible slag, especially when accompanied by a matte of high grade, which, from its high conducting qualities, has a strong tendency to chill. It is also especially influenced by the rapidity of the smelting process, a quick run with a large stream of hot slag and metal keeping a basin open where the fusion of only half the amount in the same time would chill it within a few hours. Any long stoppage is particularly detrimental, and may spoil a new basin within the first few hours.

Even under the most favorable conditions possible, a certain minimum capacity, about 20 tons in twenty-four hours, seems absolutely essential to the employment of the brick fore-hearth, and this minimum only if the matte is tolerably low grade—below 36 per cent. As this amount can usually be treated even in the smallest furnace likely to be erected, the conditions that forbid the employment of the brick fore-hearth do not often occur in the smelting of sulphide ores.

While the "chilling up" or "cutting out" of the old form of crucible in the interior of the furnace involved a costly and tedious series of operations, comprising the blowing out and cooling down of the furnace, the exterior basin can be taken down, replaced, and dried ready for work within a few hours; and it is here that the advantages of this method of practice become most apparent, as the stoppage of the blast for this short period causes little or no trouble in the furnace itself. The arrangement of the fore-hearth on wheels is a notable convenience, as the exchange can be made with great facility, and the new basin, heated to redness by a coke fire, is pushed into place between the two guiding rails, a gasket of clay being interposed between the respective abutting faces, to prevent the leakage of the liquid product. As soon as the connection is made between the main pipe and the diminutive jacket on the back plate of the fore-hearth, the clay plug with which the main orifice into the furnace was closed is pierced, and the process goes on with the slightest possible delay.*

After cooling the interior of the old fore-hearth with water, the iron plates are removed and the chilled mass broken into fragments for re-smelting.

The chill usually consists of a mixture of slag and matte, and is seldom so difficult to handle as to require the aid of blasting powder. This condition, when present, usually results from the deposition of metallic iron, which is sometimes found several inches thick and in a fine-grained, massive condition. It is best treated by exploding a cartridge of the strongest Giant powder upon it, though drilling is sometimes necessary. A ratchet-drill is used for the purpose, and a sample of borings from such a chill, analyzed for the writer by Mr. A. F. Glover, Ph.D., had the following composition:

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* The time consumed in the above operation, as taken twice under ordinary circumstances, was 18 and 21 minutes.

Sulphur	4·64	Slag	0·78
Copper	9·80	Nickel and cobalt	0·81
Iron	82·70		100.26
Carbon	1·12		
Arsenic	0·41		

This substance may be felt as a sticky, glutinous, semi-fused mass in the bottom of the basin, and is often scraped out in considerable quantities after tapping. The life of the basin is also often prolonged by a systematic chiseling out of the sides, front, and bottom, whenever empty, and a careful and energetic furnace-man will keep his fore-hearth in condition for a very long time.

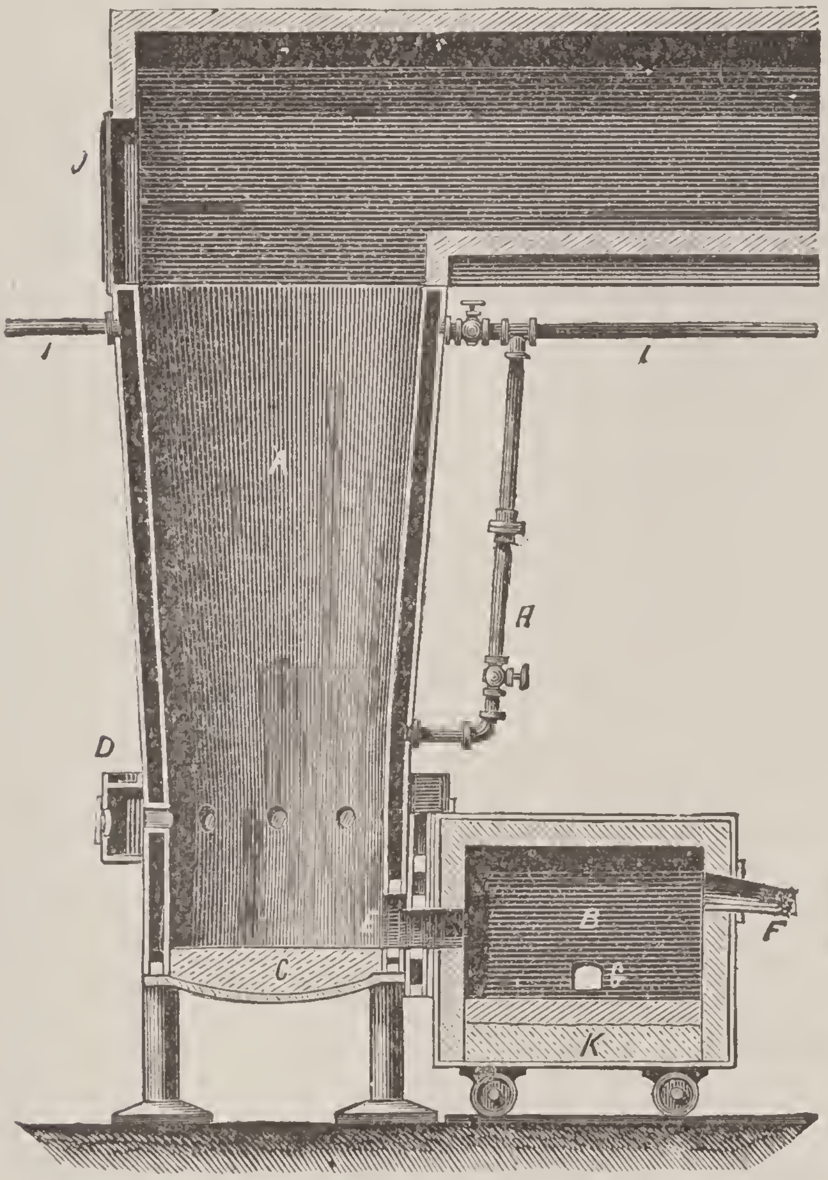
The chilling of the basin is counteracted by anything that checks the radiation of heat therefrom, and a backing of two inches of asbestos between the brick lining and iron plates is reported by Mr. Herreshoff to effect good results. The writer has used a mixture of wood ashes and crushed porous slag with good effect. The size of the basin varies according to the conditions of the case and the fancy of the metallurgist. A rectangle of 28 by 30 inches and 28 inches deep will be found convenient. It should contain from one to two tons of matte, and when inclined to chill, should be made larger at the commencement rather than under contrary conditions.

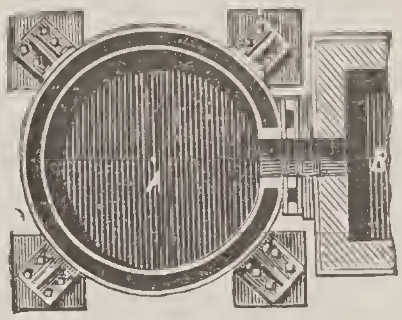
The amount of ore treated in water-jacket furnaces of the same size and with exterior basin differs greatly, according to its fusibility, the quality of fuel, and numerous local conditions. A few examples from practice will assist in forming an estimate.

Herreshoff's first water-jacket at the Laurel Hill Chemical Works, Long Island City, is shown in the accompanying illustrations.

Wendt says: "In front of the water-jacket *A*, the fore-hearth *B* is placed on wheels, and can be readily removed from the front of the furnace. The molten material flows directly from the shaft through the opening *E* into the fore-hearth. The joint between the two is readily made or severed; for it consists simply of two water-jacketed faces of iron, which are placed in contact by moving the fore-hearth *B* against the jacket *A*. The cooled iron surfaces immediately chill any matte or slag liable to run between them, and make a perfect joint between furnace and fore-hearth. The first furnace erected was round, and 48 inches in diameter. Figs. 29 and 30 illustrate its construction.

"The water-jacket *A* rests on four posts attached to the bottom by brackets, as shown in the cut. *C* is fire-clay rammed into the bottom on a supporting cast-iron plate, fastened to the furnace; *D* is the wind-box; *H* the entrance of cooling water; *I* the exit of same; *O* the charging-door; *B* the rectangular fore-hearth with fire-brick sides, top, and bottom, held by an iron plate-casing; *K* a layer of slag; *F* the cinder-lip; and *G* the tap-hole for the matte in the fore-hearth. The second furnace, built at the Laurel Hill Works, was of the same circular shape, but 60 inches diameter. Blast was furnished at the usual low pressure by a





Baker blower, and trouble was experienced by the failure of the blast to reach the center of the furnace.

"The last furnace erected, and the one now in use, shown in Figures 31, 32, and 33, is rectangular in shape, with corners rounded, and the lines between the corners slightly curved or of convex shape. The height is ten feet, width 3 feet 7 inches at the bottom, and 4 feet 7 inches at the top, by 6 feet 4 inches length at the bottom, and 7 feet 4 inches at the top. The water-jacket is exceptionally narrow, having a water-space of only 2 inches.

"Referring to the cuts, *A* is the body of the furnace; *B* a ring 2 by 2 inches, to which the plates of the water-jacket are riveted. At the top *C*, the outer plate is flanged 2 inches, and the inner plate 4 inches, and the flanges then riveted. The bottom of the furnace *E* is a disked cast-iron plate 1½ inches thick, fastened to the ring *B* by tap-bolts. This permits the dropping of the bottom if required. The legs *F* are bolted to the ring *B* on the outside of the furnace, thus not interfering with the dropping of the bottom. The hole *G* is the outlet of the furnace for both slag and matte. It is 9 inches high and 7 inches wide, and made by riveting the wrought-iron frame *H* into the shell of the furnace. The furnace is blown by 13 tuyeres, five on each side and three on the back. They are placed 26 inches above bottom plate, and are two inches in diameter."

The construction of the furnace proper is practically identical with that of the original round furnace, but the fore-hearth is considerably changed. In the round furnace (see page 201), the fore-hearth was floored with a layer of slag-wool and brick as described. A brick lining was also used. The bottom of the brick lining was some 12 inches below the outlet from the jacket. Experience proved that this bottom invariably chilled to a level with the bottom of the opening to the furnace. The cutting of the brick lining at a higher level also gave occasional trouble. Both these faults are avoided in the present construction. The former, by raising the fore-hearth on high wheels *N*, and making the floor of the bottom lining within 2 inches of a level with the bottom of the inlet *L*. The latter, by entirely casting aside fire-brick lining and depending on the circular cast-iron water-jacket *K*. The tap-hole *R* in the shaft of the furnace is used only when blowing out to tap the furnace clean, or, sometimes, for such small quantities of black copper as may be accidentally made. In the fore-hearth, the tap-hole *O* is the one commonly in use. It is made of copper, bolted to the iron body of the fore-hearth, and is water-jacketed similarly to the "Lürmann" slag tuyere of iron furnaces. The manner of operating it is also similar. *M* is the slag-spout; *W*, a brick-lined, dish-shaped movable iron cover of the fore-hearth. When smelting, the well or fore-hearth is wheeled up against the furnace, as shown in the cut, and a very small amount of wet fire-clay is placed on the iron faces surrounding the holes *G* and *L*, in order to make a tight joint between them.

In practical operation, after the furnace has been properly charged, the blast is let on. The first cinder collects in the bottom of the furnace shaft proper, and accumulates until it reaches the holes *G* and *L*. It then overflows rapidly into the fore-hearth, carrying matte with it. In a short time, the level of the molten material rises above the top of the hole *L*, and from that time onward the blast in the furnace can no longer blow out through *L*, and is completely trapped. Owing to the pressure of blast, the level of molten matte and slag in the fore-hearth is several inches above that in the furnace proper. Eventually the slag-lip *M* is reached by the cinder, which then overflows quietly. Matte is tapped periodically from the tapping-notch *O* without stopping the furnace. Matte is never allowed to accumulate until it overflows at the slag-lip, the practice being to tap at stated intervals. The notch *O* is opened by a small steel bar, and pure matte, to the amount of about 1,000 pounds, is allowed to run off. During this operation, the level of the molten slag in the fore-hearth falls, but not sufficiently to admit of blast escaping through *L*. By the simple insertion of a small clay stopper, the matte is stopped before cinder appears, thus avoiding all cinder picking. The whole process only occupies a few minutes, and is so perfect that for months a miss in tapping or closing up has not been made.

The large amount of molten slag and metal in the fore-hearth greatly facilitates a clean separation, as the slag analysis clearly shows. The high percentage of the matte, made without trouble from ironing, is entirely due to the great rapidity of the smelting. Standing at the charging-floor, the charge sinks visibly while watching it, and is exposed so short a time to the action of reducing gases that the iron is slagged before reduction, and thus ceases to be the obstacle to a rapid concentration of copper in a high-grade matte that metallurgists usually consider it.

The following data of work, done by the different sizes of furnaces, speak for themselves. The saving in fuel by the larger furnaces is apparent.

AVERAGE CHARGE PER DIEM.

	48-inch furnace.	*60-inch round furnace.*	*Rectangular furnace.*
Roasted ore..	51·7 tons	76·0 tons.	76·8 tons.
Raw fines....	4·3 "	8·0 "	13·2 "
Sand........	2·8 "	12·0 "	5·3 "
Iron slag.......		9·5 "	....
Total stock,	58·8 "	105·5 "	95·3 "
Coke.......	11·5 tons or 20 %.	17·0 tons or 16 %.	17·4 tons or 18 %.

The cost of smelting in the large Herreshoff furnace is very low. The number of employés per diem is ten, and with gas-house coke at $2.50 a ton, and repairs exceptionally low, the total cost per ton of ore cannot aggregate 80 cents, or, on the ton of 50 per cent. matte, about $10; or 1 cent per pound of copper contained.

A 42-inch jacket with five tuyeres and one-half-pound blast at Stratford, Vermont, smelts from 40 to 45 tons of roasted pyritous ore, which yields a monosilicate slag and a matte averaging 30 per cent. copper.

The 42-inch water-jacket of the late Chemical Copper Company at Phœnixville, Pa., with tuyeres and blast as in last example, smelted from 35 to 50 tons daily, according to the character of the charge.

In Butte City, Montana, a 48-inch wrought-iron jacket, with six 2-inch tuyeres and five-eighths of a pound blast, smelted from 60 to 65 tons daily of calcined pyritic concentrates, largely in a fine condition. In this instance, the richness of the ore, combined with a quite thorough calcination, gave such a high grade of matte as to render the employment of an external fore-hearth an impossibility, in consequence of the rapid chilling of the metal, which soon closed the tap-hole effectually, and in thirty-six hours reduced the basin to too small a size, which difficulty has been obviated by substituting an "Orford" fore-hearth, with automatic tap, to be described later.

These are average results, the fuel in all cases being coke of good quality, with from 12 to 15 per cent. ash, and the ores rather basic than siliceous. The performance of the same type of furnaces when producing pig-copper from oxidized ores will be noticed later.

By inexperienced metallurgists, the general construction and arrangement of the water-jacket furnace may safely be left to the manufacturers, several of whom have had a wide experience in this matter. The quality of the material to be used has already been noticed, nor is it good management to economize in this particular.

The water-jackets are usually made from sheets of iron or steel, rolled to the proper size, to avoid unnecessary riveting, and the rivet-heads on the inside of the shaft should project as little as possible, to avoid burning off.

The closing in of the water-space at top and bottom is effected by bending over and riveting the sheets together, or better, by the introduction of a circular ring of 2-inch square wrought-iron, forming a solid frame, through which the rivets pass, holding the iron plates firmly in place. A similar arrangement of cast blocks is provided for the tuyere openings, the castings having a central orifice of the proper size for the tuyere and a circular row of perforations for the passage of the rivets. The tuyere openings are usually equally spaced, and from 18 to 20 inches from center to center.

A convenient arrangement for the distribution of the wind consists in a cast-iron wind-box surrounding the furnace and in air-tight communication with the tuyere openings, the blast entering the former from the main wind-pipe. In the Herreshoff furnace, the tuyeres are rendered easily accessible by a circular hinged plate opposite each, and provided with an eye-glass. In other cases, the tuyeres consist merely of galvanized iron pipes, connected with properly placed branches from the circular main blast-pipe surrounding the furnace, though independent of the latter.*

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* Professor Richards, of the Massachusetts Institute of Technology, has improved on the above by the use of an ordinary steam-pipe of the proper size, which is ground slightly tapering, to make a tight joint in the tuyere opening, while a tee at the opposite end provides for the connection with the overhead blast-pipe, as well as for a glass eye-piece.

The connection between blast-pipe and tuyere-pipe is usually made by the so-called tuyere-bag, consisting merely of a light duck hose, soaked in alum-water to render it uninflammable and impervious to the wind.

The height of the furnace depends on the quality of ore and fuel, as well as the nature of the process; refractory, siliceous ores, and dense, strong coke or charcoal permitting and requiring the employment of a much higher furnace than the opposite conditions. As the greater number of water-jackets running on sulphide ores in this country are favored with a basic and easily fusible charge, any height above 10 feet—from tuyeres to charge-door—is rarely met with; and even when smelting more infusible material, the danger of reducing metallic iron and the general unmanageability of a high furnace would render of doubtful value any increase of the height beyond 14 feet.

The arrangement of the upper portion of the furnace will depend principally upon the ultimate disposal of the smoke and fumes. The simplest and cheapest plan consists of a strong sheet-iron stack, lined with a single thickness of fire-brick, and erected upon the same columns that support the jacket itself.

Such an arrangement could hardly be permanent, as the flue-dust from any material worth smelting should be sufficiently valuable to pay for saving. As this usually involves the leading of the furnace gases down to the level of the ground, it is customary to effect this by means of a so-called "down-take," consisting of a vertical or inclined flue, leading from the furnace at a point above the charging-door to the entrance of the condensation-chambers or subterranean flue system.

The charging-door should be proportionate in size to the furnace, and should open from 12 to 18 inches above the charging-platform, to insure the proper feeding of the furnace; for dishonest laborers find it more convenient merely to push the ore into an opening level with the floor than to scatter it in the careful and systematic manner so essential to the regular working of the furnace.

In cases where the occurrence of zinc-blende or even galena in the ore renders the formation of wall-accretions a matter of probability, it will greatly facilitate their removal to have the charging-door in the sloping roof-shaped housings above the furnace shaft, as long bars can thus be introduced with ease. This opening should be provided with a close-fitting door, and an easy-working iron damper should be placed in some accessible part of the chimney or down-take, a great diminution of flue-dust being observable when the rapidity of the chimney draught is so checked that the fumes are barely carried away.

The difference is obvious between the conditions in cupola practice, where the draught merely serves to remove the fumes produced, while the combustion results entirely from the blast below, and reverberatory work, where the burning of the fuel, and consequently the temperature, depend solely on the draught produced in the chimney.

The ordinary shape of the American water-jacket is that of the frustum of an inverted cone or pyramid, the upper diameter being from 8 to 12 inches greater than the lower, while the use of boshes is very rare, owing to the causes already mentioned as influencing the height of ordinary furnaces.

A slight bosh would be quite in place in smelting refractory ores, and is used to advantage in the Bell furnace at Butte City, where a very siliceous charge is smelted with poor charcoal.

Where condensation-chambers or long flues exist, the size of the chimney seldom stands in any exact relation to the requirements of a single cupola; but where no such passages are interposed, any data of the capacity of stack necessary for a furnace of a given size are useful, especially in the case of blast-furnaces, where, owing to reasons already mentioned, the ordinary rules governing the subject cannot be applied.

For a single circular chimney, either directly above the jacket-shaft, or communicating with the same by a non-descending flue, experience shows that the ratio between the diameter of the chimney and the furnace diameter at the tuyeres should be about as 2 to 3. Thus, a 36-inch furnace requires a 24-inch stack; a 60-inch furnace, a 40-inch stack, etc.*

Any considerable lessening of this ratio is likely to interfere with the draught and give rise to an annoying and injurious escape of gas from the charging-door.

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* If the sizes of chimneys here given seem unnecessarily large, it must be remembered that, when in good condition, the blast-furnace is so cold on top as to permit the introduction of the naked hand, and consequently, the temperature of the column of air in the stack is so low as to cause but little difference in weight between the interior and the exterior.

The following measurements have been taken from stacks in various works in the United States, being selected from a considerable number:

Diameter of furnace at tuyeres, in inches.	Diameter of stack, in inches.	Resulting draught.
42	30	Excellent.
40	24	Fair.
36	20	Feeble.
48	30	Feeble.
48	36	Excellent.

The last two measurements refer to the same furnace, the draught being so poor with the smaller chimney as to require its enlarging.

While an elevation of a few feet above the roof is sufficient to carry off the fumes, safety demands that cupola stacks should have such a height that, during the blowing in and out of the furnace, the sparks and burning fragments of fuel that are then projected in considerable quantities shall be carried to a proper elevation.

The water-jacket furnaces employed for the fusion of oxidized ores do not differ in any essential particulars from those just described. Being used principally in Arizona, New Mexico, and other distant parts of the country, where mechanics' and masons' labor as well as fire-brick and similar refractory materials are very dear, these furnaces are so arranged as to be almost entirely independent of those sources of expense after their first erection. This is effected by the use of heavy sheet-iron housings for that portion of the structure above the upper edge of the jacket, upon which fits a chimney of the same material, the iron in every case being protected by a 4-inch thickness of fire-brick, properly shaped to form the circle.

Instead of the exterior crucible or fore-hearth just described, the Western water-jackets are for the most part provided with a cylinder of boiler iron, which projects downward below the jacket, forming an extension of the same, and provided with a falling bottom, consisting of two hinged iron doors, which, when supported in place by an iron bar, form a foundation for the support of the quartz bottom, while the cylinder referred to is lined with fire-brick, thus forming an interior crucible, the full size of the furnace, and extending from 16 to 24 inches below the tuyere openings.

In rare instances, the water-jacket is continued to the extreme bottom of the crucible; but when handling a product so inclined to chill as is metallic copper, the arrangement just described is probably the best.

The peculiarly favorable composition of the Copper Queen and many other of our Southern carbonate ores, being entirely oxidized, and containing an ample proportion of iron and lime, has permitted the employment of low, cheap furnaces, as well as the fusion of an unusual amount of ore in proportion to their size. And it is to this favorable condition of affairs, rather than to any inherent virtue on the part of the furnaces used, that the extraordinarily long and successful campaigns of the Copper Queen and neighboring furnaces must be attributed.*

The following figures, taken from Mr. Douglas's paper, are average results of regular work:

The Copper Queen smelter contains two 36-inch circular wrought-iron jackets, each of which puts through from 45 to 50 tons of ore daily, flux being seldom required. To these a 150-ton water-jacket has lately been added.

The very fusible ore of the Old Globe mine (Arizona) is smelted in a 3-foot furnace at the rate of 55 tons daily, and even this extraordinary result has been exceeded by the United Verde furnace.

In nearly all cases, a No. 4½ or 5 Baker blower is used, which, at from 100 to 115 revolutions, supplies from 5 to 7 tuyeres with wind at a pressure of from 10 to 12 ounces.

The size of the tuyeres is very variable, 3 inches being the average diameter, although the Copper Queen management has found a decided advantage, both in capacity and in freedom of the slag from copper, by increasing this measurement to 5 inches.

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* As the water-jacket furnace has had its principal development in the smelting of oxidized ores, and as its whole construction and management are peculiarly American, it seems proper to describe the same with some minuteness, taking as a type the plant of the Copper Queen mine, of Arizona, where, under the direction of Mr. Lewis Williams, it has been thoroughly adapted to the surrounding conditions. Ample use will be made of the valuable paper by Mr. James Douglas, entitled "The Cupola Smelting of Copper in Arizona," which was written for the United States Geological Survey, Albert Williams, Jr., editor.

About 1,000 fire-bricks are required, on the erection of the furnace, for the lining of the portion above the jacket, and for the crucible. None is subsequently used, as the upper lining lasts indefinitely, while the crucible is kept from burning out by the introduction of siliceous or clayey ore through the tuyeres, whenever a too hot basic slag has thinned its walls and bottom beyond the normal standard. Any indications of chilling up are at once counteracted by a slight addition of fuel, and by permitting the flame to blow through the tap-hole and metal opening.

These orifices, provided with cast spouts, are situated respectively 10 and 24 inches below the tuyere openings, the latter being at the very bottom of the crucible.

They are closed by inch bottom-plates, perforated with a large opening, the slag flow being cooled by water. Even without such cooling, these plates are found to possess decided advantages over the ordinary brick-work openings.

The cooling water is introduced into the jacket through four 1½-inch pipes, at some distance above its lower edge, and should be deflected at right angles from its horizontal course, experience having shown that its constant spouting against the hot inner iron plate causes a rapid perforation of the same. Where lime salts are present, it should never be allowed to reach a high temperature, on account of the formation of scale.

The small quantity of water required after the furnace has reached its full burden, compared with what is necessary during the operations of blowing in and out, although usually attributed to the formation of a coating of slag on the interior of the jacket, is, in the author's opinion, due rather to transference of the point of greatest heat to the center of the shaft. This arises from the formation of slag noses around the orifices of the tuyeres, by which the blast is conducted away from the walls, which are thus left comparatively cool.

The large amount of water necessary to cool the Copper Queen jackets—some 40,000 gallons daily—suggests some relation between that circumstance and the great size of the water-space in the jackets there used, being 9 inches wide at the bottom and 4½ inches at the top. In the Herreshoff furnace, and three other jackets employed by the writer, the water-space has not exceeded 2 inches, and though the diameter of the furnaces was considerably greater than that of the Copper Queen—all of them being above 42 inches—a single 2-inch pipe under a slight head was quite sufficient to supply the cooling water. This is a point well worth examining, especially where water is scarce and costly, as in our Southern carbonate districts. It is possible that the impure nature of the Copper Queen water, requiring the removal through hand-holes of the calcareous deposit every five weeks, may necessitate the broad water-space used.

THE MANAGEMENT OF WATER-JACKET FURNACES.

The slight amount of brick-work in this type of furnace requires but a single night's drying, a brisk fire being maintained in the crucible, upon which by four A.M. enough coke should be thrown to fill the furnace some 30 inches above the tuyeres, care being taken to secure an ample circulation of water through the jacket before the initial fire is kindled.

By seven A.M. the coke should be in full glow, the combustion at first proceeding slowly, as its sole air supply comes from the open tap-hole and slag flow, but progressing very rapidly after once attaining the tuyere level. Instead of turning on a light blast at this point, and gradually filling the furnace with alternate layers of charge and fuel, a much safer and more convenient method consists in filling the furnace level with the charge-door before using the blast at all, by which means all excessive heat is avoided.*

A single exception may be made in favor of a small independent tuyere, consisting of a 2-inch pipe connected with the blast-pipe by a canvas hose, which should be thrust through the breast opening in an oblique downward direction, the tuyere orifices being tightly plugged, so that the flame may issue from the tap-hole—if such exist—while the whole interior of the crucible is brought to a white heat. Fresh coke being added until it stands some 30 inches above the tuyeres—which are left open—a charge of basic, easily fusible slag is given, alternating with coke charges in the ratio of three pounds of charge to one of fuel for the first three charges, after which the ratio may be changed to 4 to 1, while one-fourth of the slag is replaced by the regular ore mixture; this may be continued for two or three charges, when the relation of charge to fuel and of ore to slag should be again raised, reaching under ordinary circumstances a burden of from 5½ to 1 by the time the level of the charging-door is attained. Then, and not before, a light blast is turned on, and the rapidly sinking column replaced by constant charges of ore and fuel, the proportions between the same being regulated by the condition of the crucible and the appearance of the tuyeres and slag.

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* The method of blowing-in here described, though practiced for many years by the writer, has been recommended by a correspondent of the *Engineering and Mining Journal*, whose experience confirms the author's views.

It is a matter of great importance to make a good start, and especially where the brick fore-hearth is used to insure a rapid current of hot, liquid slag. A delay of a very few moments while the basin is first filling, or a thick, cold slag, will cause the formation of a chill that will very probably fill up the fore-hearth to half its extent.

To avoid such a mishap, particular attention should be devoted to the selection of the slag used for blowing in, and no fear need exist of its being too basic, provided the latter quality is due to oxide of iron and not to an excess of lime. For this purpose, nothing can excel the so-called "metal slag," produced from the fusion of roasted matte, with the addition of just sufficient silica to produce a proper slag. While the percentage of SiO_2 in this material often falls below 22 per cent., no fear need be entertained but what it will become sufficiently acid in its passage through the furnace to obviate any of those troubles that arise from the production of a subsilicate during ore smelting. The clay mortar and lining of the crucible and fore-hearth, the dirt that becomes mixed with it during transportation, and, above all, the ashes from the fuel, will be found quite sufficient to neutralize any excess of base. By the gradual substitution of the ore-charge for this metal slag, a less and less basic slag is produced, until the normal charge is reached, and the thin, blood-red, smoking, acrid slag that first appears is replaced by a white, steady, and slightly viscid stream, almost free from smoke—unless unusually basic—and satisfactory to the experienced eye. In the water-jacket furnace this condition and the normal charge of ore and fuel should be reached within twelve hours, unless some drawback occurs, due to a neglect of some of the precautions already mentioned, or to a too rapid increase of blast or of the ore burden. Where sulphide ores are treated, the initial charge should contain enough unroasted ore to produce a matte below 35 per cent. copper for the first two or three tappings. By this means the fore-hearth is heated to the proper point, and the chilling due to too rich a matte at too low a temperature is avoided.

Where oxidized ores are treated, and an interior crucible used, the means already mentioned will secure a proper temperature and prevent chilling.

But the exterior "steep" crucible, although previously heated to redness by a coke fire, is pretty likely to become partially filled by a so-called "skull," or chill of metal, which lines the sides and bottom to the thickness of several inches, rendering it impossible to tap and difficult to ladle. This chill should be allowed to form until the furnace has reached its normal condition, and slag and metal are flowing freely, when it may be removed by means of pointed bars and an overhead tackle. It leaves a glowing hot crucible, which, after a little repairing, is allowed to fill with the hot products, and is then carefully covered with a mixture of fine charcoal and ashes, by which means it is kept from chilling for a long period.

The Copper Queen, and nearly all water-jacket furnaces running on oxidized ores, are provided with an interior crucible, in which both slag and copper collect, the former being tapped into pots every six minutes, while the metal is run through the 14-inch lower spout every half-hour into iron molds, mounted on wheels, and which hold about 250 pounds.*

The tapping is accomplished with great ease, using only a light, pointed iron rod, the aid of the sledge being scarcely ever required. As the openings for both slag and metal are made in jacketed copper plates, they never become too large for easy plugging, and are stopped by a minute ball of clay.

Where the ores are easily fusible and produce no wall accretions, and, above all, where constant supervision and extreme care are bestowed upon the smelting process, as at the Copper Queen, the length of the campaigns seems dependent only upon the life of the machinery and jacket, although it is found necessary at these works to burn down the furnace to the tuyeres every five weeks in order to remove the calcareous sediment from the water-space. Still, as the crucible is not cooled off, and as the furnace resumes its normal condition as soon as refilled, these brief interruptions can scarcely be considered as terminations of a campaign.

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*In almost every case where Arizona practice is referred to, Mr. J. Douglas's paper has been consulted either for direct information or verification.

The conditions at these works are peculiarly favorable, no extraneous flux being required when proper proportions of ore can be obtained from the different levels, where calcareous, ferruginous, and siliceous mixtures of carbonates and oxides of copper are all represented. The following analysis of the average slag for several months is taken from the report already mentioned:

	Per cent.		Per cent.
Silica	26·64	Magnesia	0·20
Protoxide of iron	42·60	Alumina	15·40
Manganese	0·30	Alkalies and loss	4·85
Zinc	0·50		
Lime	9·51		100·00

The average slag assays from the smelting of oxidized ores in water-jackets may be estimated at 1·75 per cent. copper, consisting of about equal proportions of oxidized and combined copper, and of minute metallic globules.

A rectangular water-jacket, built for the Detroit Smelting-Works, Clifton, Arizona, by Mr. C. Henrich, presents certain interesting points for comparison. The cross-section at the tuyeres is 33 by 66 inches, while 10 inches above the same a bosh is begun, so that 30 inches above the tuyeres the cross-section is enlarged to 45 by 78 inches. The four lower cast-iron jackets terminate at this point, where they are surmounted by four others, which still diverge slightly, so that at their upper surface, 7 feet 6 inches above the tuyeres, the furnace has an inside section of 54 by 87 inches, which is retained to the charging-door, 10 feet 6 inches above the tuyeres. The slag-tap is 6 inches below the latter, and the crucible is 14 inches deep, lined with brick, and provided with a drop bottom. There are fourteen 2¾-inch tuyeres, five on each side and two at each end, receiving a blast of 10 ounces from two No. 4½ Baker blowers, making 115 revolutions. The object of the bosh is to increase the reducing action, with the view of obtaining cleaner slags, a result that is claimed to have been obtained, the slags from this furnace assaying 0·4 per cent. lower than from the small perpendicular furnaces. There was also a saving in coke of 8 per cent.

The ores smelted are mixtures of carbonates and oxides, and, being slightly too siliceous, require the addition of a small proportion of limestone and iron to produce the following slag, an average of several weeks, as determined by Mr. S. James, Jr.:

	Per cent.		Per cent.
Silica	34·34	Alumina	11·80
Protoxide of iron	32·27	Alkalies and loss, etc.	3·64
Manganese	6·24		
Lime	10·13		100·72
Magnesia	2·30		

An elliptical furnace, provided with sectional cast-iron jackets, forming a bosh 29 inches high immediately above the tuyere level, has been in use for many years for treating the slags resulting from the fusion of the Lake Superior metallic "mineral" in reverberatory furnaces, previous to the refining operation, which is merely a later stage of the same fusion.* (See section on Refining.)

The cupola referred to is a modification of McKenzie's pig-iron cupola, and has, in place of distinct tuyere-openings, a five-eighth inch slot encircling the entire furnace, just below the water-bosh. Below the tuyeres is a 34-inch deep crucible, nearly the full size of the furnace, and closed by a drop-bottom, protected by a few inches of sand. The water-bosh consists of curved sections of cast-iron, fitted closely together, and five-eighths of an inch thick on the inner and lower sides, while the external and superior sides have only half an inch of metal. This bosh is 22 inches high, and is kept cool by a three-quarter inch supply-pipe, furnishing 25 gallons of water a minute.

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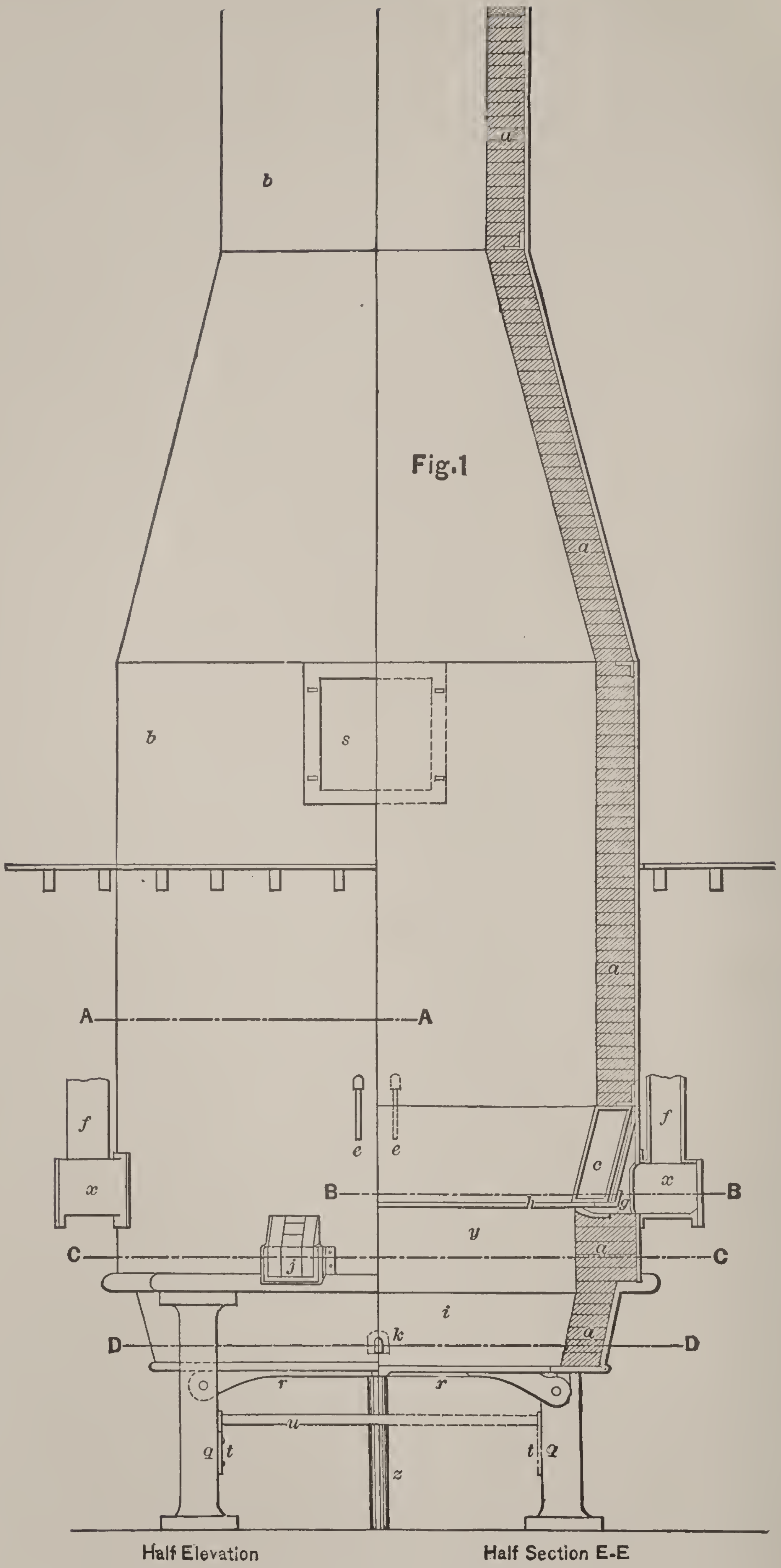
* The measurements and other details of these slag cupolas may be seen in the accompanying illustrations, which first appeared in connection with Prof. T. Egleston's paper on "Copper Refining in the United States," published in the *Transactions of the American Institute of Mining Engineers*, Vol. IX., of which use is also made in the following descriptions, as verifying the writer's own observations.

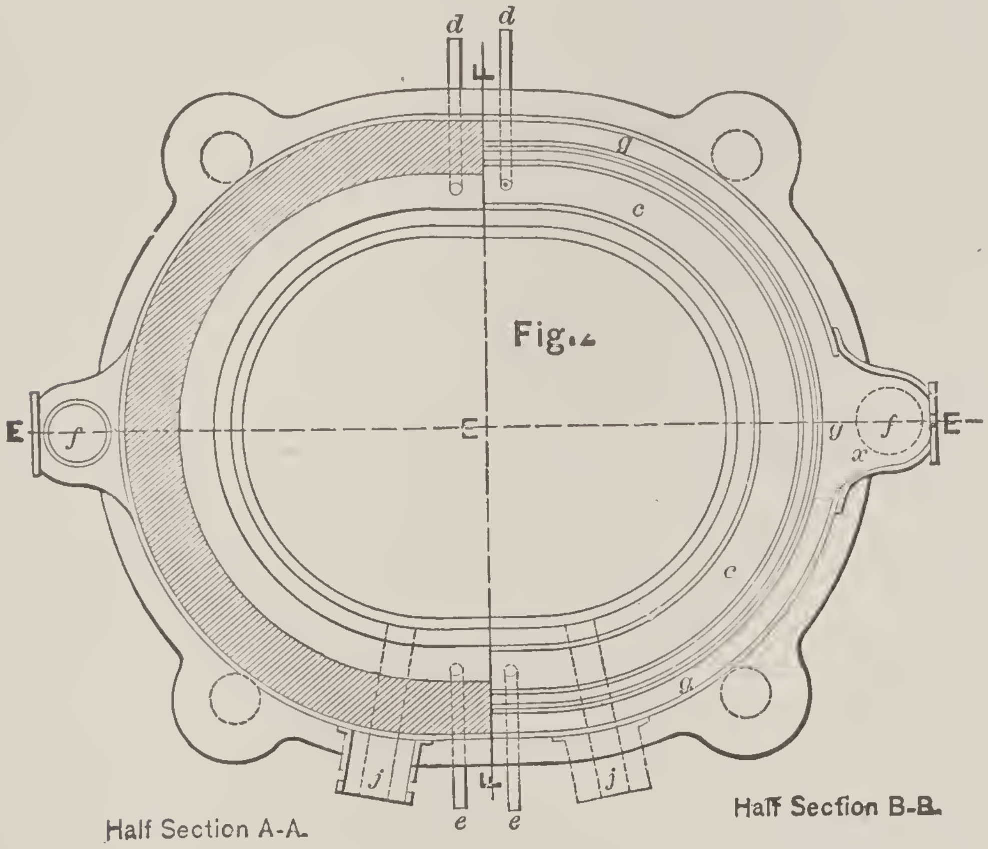


SLAG CUPOLA—ELEVATION.

The cupola is 7 feet 6 inches in height from tuyere level to charging-door, and has a greater axis of 7 feet and a smaller one of 4 feet 9 inches.

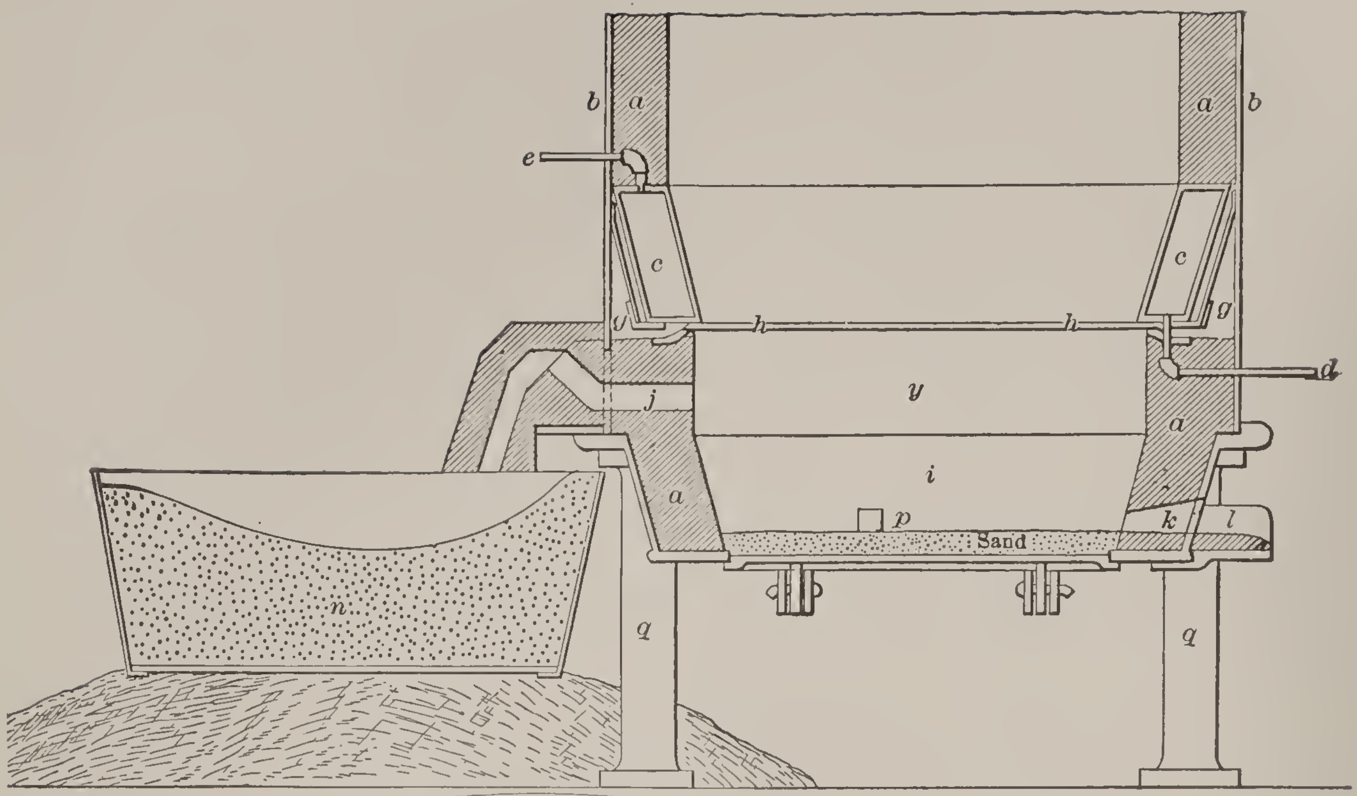
A peculiar inverted siphon arrangement for the trapping of the blast during the continuous slag flow will be noticed by reference to the illustrations. As even during the ten-hour campaigns made by these furnaces (owing to the small lots of slag belonging to separate mines), two of these slag flows are pretty thoroughly used up, they will hardly be likely to come into general use.

The material smelted is a siliceous slag from the reverberatory furnaces, carrying some 15 per cent. of copper and over 40 per cent. of silica. About 20 tons are smelted in ten hours, using anthracite as fuel, brief attempts to use coke having resulted in an increase in the richness of the final slag. About 1 pound of coal is used to smelt 3 pounds of slag, probably the highest consumption of fuel in the United States. Blast is furnished by a No. $5\frac{1}{2}$ Baker blower at a pressure of 10 ounces per square inch. The metallic copper produced is impure, containing some 5 per cent. of iron and half of one per cent. of sulphur, the latter coming from the fuel; while the large amount of iron present is also due to the powerful reducing action of the anthracite, which



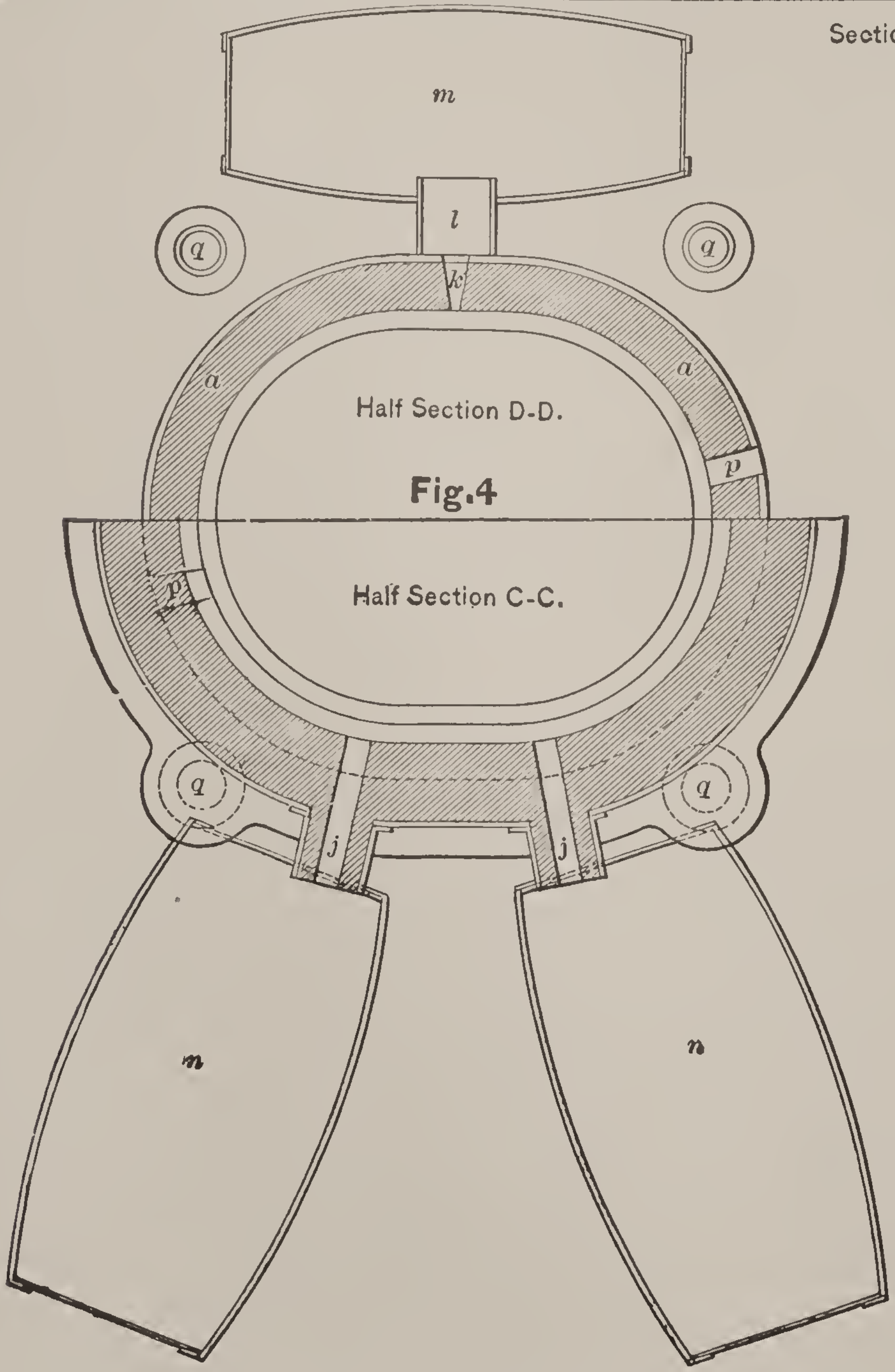
SLAG CUPOLA—PLAN.

Fig.3



Section F-F.

Fig.4



SLAG CUPOLA.

seems necessary to decompose the silicate of copper present in the slags.*

On account of the daily blowing out of these furnaces for the reasons already alluded to, and the necessity of maintaining a strict separation of the material belonging to the various mines, the furnace bottom and sides, up to the water-bosh, are torn out and renewed every run. Although this practice is made necessary in order to obtain every scrap of copper-bearing material belonging to each special campaign, still the deeply eaten and worn condition of the brick lining shows that metallurgical reasons also exist for this laborious and expensive custom.

In common with the Arizona furnace managers, those at Houghton (Lake Superior) have also found that the slag flowing from the furnace contains an appreciable amount of copper in the shape of fine beads, technically denominated "prills." In both localities, recourse has been had to an independent fore-hearth, consisting of a rectangular iron box, in which the metal settles during the slow progress of the slag.

This fore-hearth at Lake Superior is lined with a mixture of clay and sand, and is 6 feet 6 inches long, 3 feet 6 inches wide at the middle, and 2 feet 9 inches at each end, being slightly oval in shape. A considerable quantity of scrap-iron is placed in it at the beginning of the run, with the view of reducing to metal the oxidized copper contents of the slag, and is successful to the extent of saving some 10 per cent. more than when omitted. A cake of from 150 to 200 pounds is usually obtained from the daily run, from a charge of 20 tons of slag, yielding 15 per cent. of copper, the fore-hearth therefore saving about 2½ per cent. of the entire metallic contents. The slag from these cupolas is very clean, considering the grade of the product, and is reported to average below 0·75 per cent., rarely reaching one per cent. It is rejected as worthless.

The fore-hearth in use at the Copper Queen furnaces for catching the entangled metallic shot is a rectangular box, made of four cast plates and mounted on small wheels. Its inside dimensions are 4 by 2½ feet, and 30 inches deep. It is lined with a mortar of clayey ore, and in an average life of 36 hours yields about 150 pounds of copper, besides the metal entangled in the chilled slag with which it becomes filled.*

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* The formation of so large an amount of silicate of copper during the primary smelting in reverberatories might possibly be prevented by the addition of fine coal to the charge, although, of course, its fusibility would be somewhat lessened thereby.

The following table, compiled from the articles already acknowledged and from the writer's own notes, exhibits comparatively certain points of interest pertaining to the smelting of oxidized ores in water-jacketed cupolas:

NAME OF SMELTING COMPANY.	Area of furnace at tuyeres in sq. ft.	Number of tuyeres.	Total tuyere area in sq. ins.	Pressure of blast in oz. per sq. in.	Pounds of ore to 1 pound of fuel.	Pounds of ore to 1 pound of charge.	Tons of ore per 24 hours.	Tons of charge per 24 hours.
Detroit Refining Works (Houghton, Mich.)......	24¾	Continuous tuyere.	134	10	3·00	4¼	40·00	56·00
Copper Queen............	7	6	75	10	5·90	5·90	47·00	47·00
Old Dominion............	7	6	64	10	....	....	.....	.....
Detr'it C'r Co. large furnace	11·8	14	83	12	6·55	7·02	79·15	86·65
" " small "	7	6	58	10	5·55	6·00	45·00	48·60
Ariz'na C'r Co. large furnace	12·5	6	64	10	....	7·75	.....	75·00
" " small "	7	6	58	10	....	7·75	.....	55·00
United Verde.............	7	6	58	10	....	5·00	52·00	.....

A noteworthy feature of the Arizona cupola practice is the purity of the product, averaging between 97 and 98 per cent. of metallic copper. Its freedom from injurious substances is, of course, due to the quality of the ore; but the low percentage in iron of pig-copper, produced in many instances from highly ferruginous ores, and by a process of reduction so powerful that only traces of oxidized copper remain in the slag, must be attributed to the rapidity of the fusion. This in its turn results principally from the volume and pressure of the blast and suitability of the fuel, which consists in most cases of a coke of tolerably good quality from Trinidad, Colorado, or San Pedro, New Mexico, the latter containing less ash, but being of a more friable nature. With Connellsville (Pennsylvania) coke, a slightly higher ratio of ore to fuel is obtained, and a patent Cardiff coke gives the best results of all. The average contents of ash in the Trinidad, San Pedro, and Connellsville cokes is reported respectively at 14·6, 13·2, and 11·6 per cent. Sufficient sulphur is present in nearly all the Arizona carbonate ores to form a small proportion of matte, which in many cases is simply thrown back into the furnace without further treatment, while other companies more sensibly sack and ship it East as a separate product. It varies between a high blue-metal and a low white-metal from 60 to 66 per cent., and could be advantageously roasted twice in heaps and mixed with the ore-charge in small quantities.

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* See section on Brick Cupolas for other varieties of fore-hearth.

At the Copper Queen Works, a kiln is used for the roasting of whatever matte may be produced, though its occurrence is very irregular. Sometimes, for several successive days, as much as 1,500 pounds per day will be made, and is easily separated from the bars of metal on which it floats. At other times, for months together, not a trace of matte will occur; nor does the depth from which the ore is mined have any influence on its production, for the sulphide ores which cause it are found fully as abundantly in the upper as in the lower levels.

Even the freedom from all concentration and calcination processes does not entirely relieve the Arizona smelters from that great curse of blast-furnace work—the occurrence of a considerable proportion of fine ore. The clayey and friable nature of many of the ferruginous and calcareous carbonates favors the formation of fines, which, aside from the heavy loss entailed by their escape through the stack, clog the furnace, obstruct the blast, and, being sifted down between the coarser lumps of ore and fuel, reach the smelting zone in a cold and unprepared condition, causing the chilling of the crucible and the growth of long noses from each tuyere, which may meet in the middle, forming a central core of semi-fused material that may necessitate the termination of the campaign. The loss in flue-dust is partially remedied at the Copper Queen and one or two other furnaces by the construction of flues and dust-chambers. The clayey nature of most of the fines, and the hot and dry climate, assist the process of bricking these fines, although in certain cases an addition of milk of lime is found necessary to bind the particles together with sufficient firmness.

Most of the ores being of a basic and decomposed nature, a very small amount of mechanical preparation for the furnace is demanded. At most works, the ore is merely passed through a jaw-breaker, set to a size of from 2 to 3 inches.

For the reasons already enumerated, a height of the furnace from tuyere to charging-door of more than seven feet is rarely met with in this particular practice.

The use of round or oval furnaces of the water-jacket type has become almost universal in copper smelting, probably owing to their ease of construction. A long and varied trial by the writer of almost every style and size of cupola was so indubitably in favor of the rectangular or elongated oval form, and more especially of very much larger furnaces than any yet described, both for economical reasons and for ease and simplicity of management, that the comparative want of success in certain reported cases is apparently attributable to other causes than a mere incapacity on the part of the furnace to fulfill all expectations based on comparative calculations. The new furnace built by the Detroit Smelting Company, of Michigan, which is much larger than the older ones, is entirely satisfactory, and works with increased economy, reaching a result fully equal to its theoretical capacity.

The great size of the furnaces preferred by the writer has, until quite recently, prevented the employment of water-jackets, so that their description and discussion must be deferred to the section on Brick Blast-Furnaces, but since the success of quite large, oval, wrought-iron jacketed cupolas has been assured, there can be but rare instances where brick furnaces would be preferred.

The life of a water-jacket in constant use depends so entirely upon its treatment and upon the quality of the feed-water that it is impossible to fix any exact limit for it. Cast-iron jackets may last from one to four years, though sometimes cracking in a few days, while wrought shells have been run almost constantly for six years without any considerable repairs. Under the most favorable conditions, and in the lack of more extended experience, five years may be assumed as the duration of a wrought-iron jacket in constant use.

Estimates of cost would be superfluous, the manufacturer's price-lists and cost of boiler work and piping supplying all needful bases for calculation.

Owing to the comparatively recent introduction of water-jacketed furnaces with improved arrangements for crucible and fire-hearth, there is a great lack of accurate information on the sub-

Number.	Character of ore.	Percentage of ore.	Character of flux.	Pressure of blast in ounces per square inch.	Character of fuel.	Per cent. of ash of fuel.	Area of furnace at tuyeres in square feet.	Number of tuyeres.	Area of tuyeres in square inches.	Height of furnace.	Ore per 24 hours.		Charge per 24 hours.		Composition of product.			Composition of slag.				
											Tons.	Ratio to fuel.	Tons.	Ratio to fuel.	S.	Cu.	Fe.	SiO_2	FeO.	CaO.	Al_2O_3	Cu.
1	Calcined pyrites..	3½	Gravel...... ...	12	Connellsville coke	12	9·6	6	42	8	56	7·9	64·4	9·1	28·5	33	*31	23	63·3	1·7	7·9	0·3
2	Western carbonates............	21	Metal slag... ..	......	....	....	.....	..	....	...	43	6·8	53¾	8·5	0·6	97·4	*2	38	55 7	0·8	?	1·65
3	Calcined matte...	33	Siliceous carbonates.......	......	....	....	... ..	..	....	....	48½	7·3	60	9·1	?	98·2	?	31	59 6	?	?	1·47
4	Calcined matte...	...	....	6	... .	....	. ....	..	....	....	27·6	6·6	34·5	8·2	1·1	96·2	2·4	34	?	?	?	0·96
5	Carbonates and oxides.........	14½	Cupriferous, hematite.....	9	Trinidad coke...	14½	8·25	5	2·9½	6½	31	4·7	41¼	6·25	0·5	97·6	1·7	36·5	40	*13	6·7	1·7
6	Calcined pyritic concentrates...	18	Siliceous carbonates......	10	Connellsville coke	12	12·5	6	42	7	66	6·3	71¼	6·8	*28	61·5	10	29	63	?	?	1·1
7	Pyrites fines, unroasted.........	7½	Metal slag.. ...	10	Gas-coke..... ...	16	9·6	6	42	7¼	38	3·5	66	6	36	19½	53¼	46	47·6	?	2·2	0·11

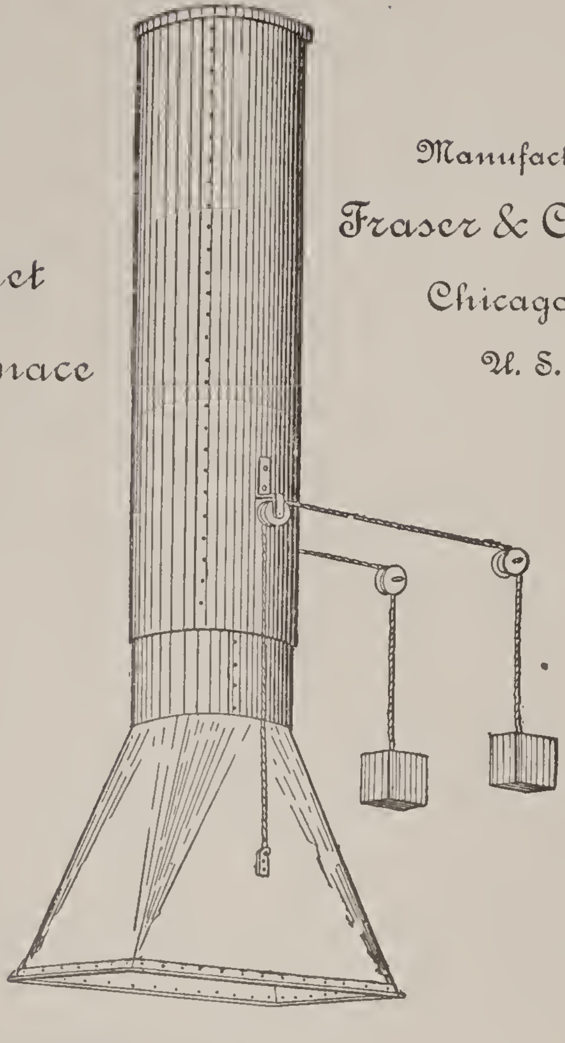
NOTE.—Numbers 1, 2, 3, and 4 are from the same furnace.

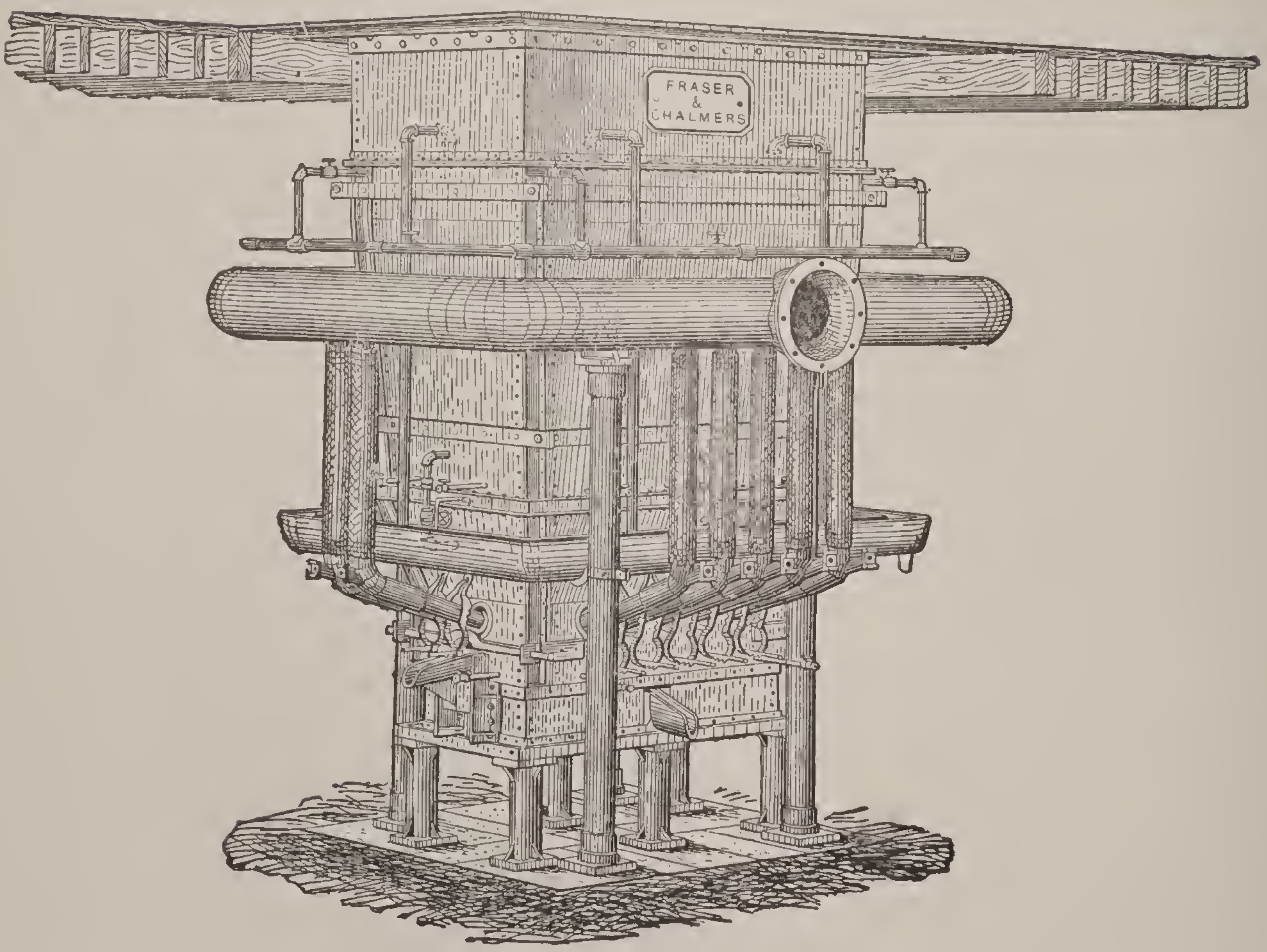
* These figures have been estimated. In all other cases, they are the result of actual determinations.

The author is indebted to his various chemists and assistants at the different furnaces for most of the chemical results here given.

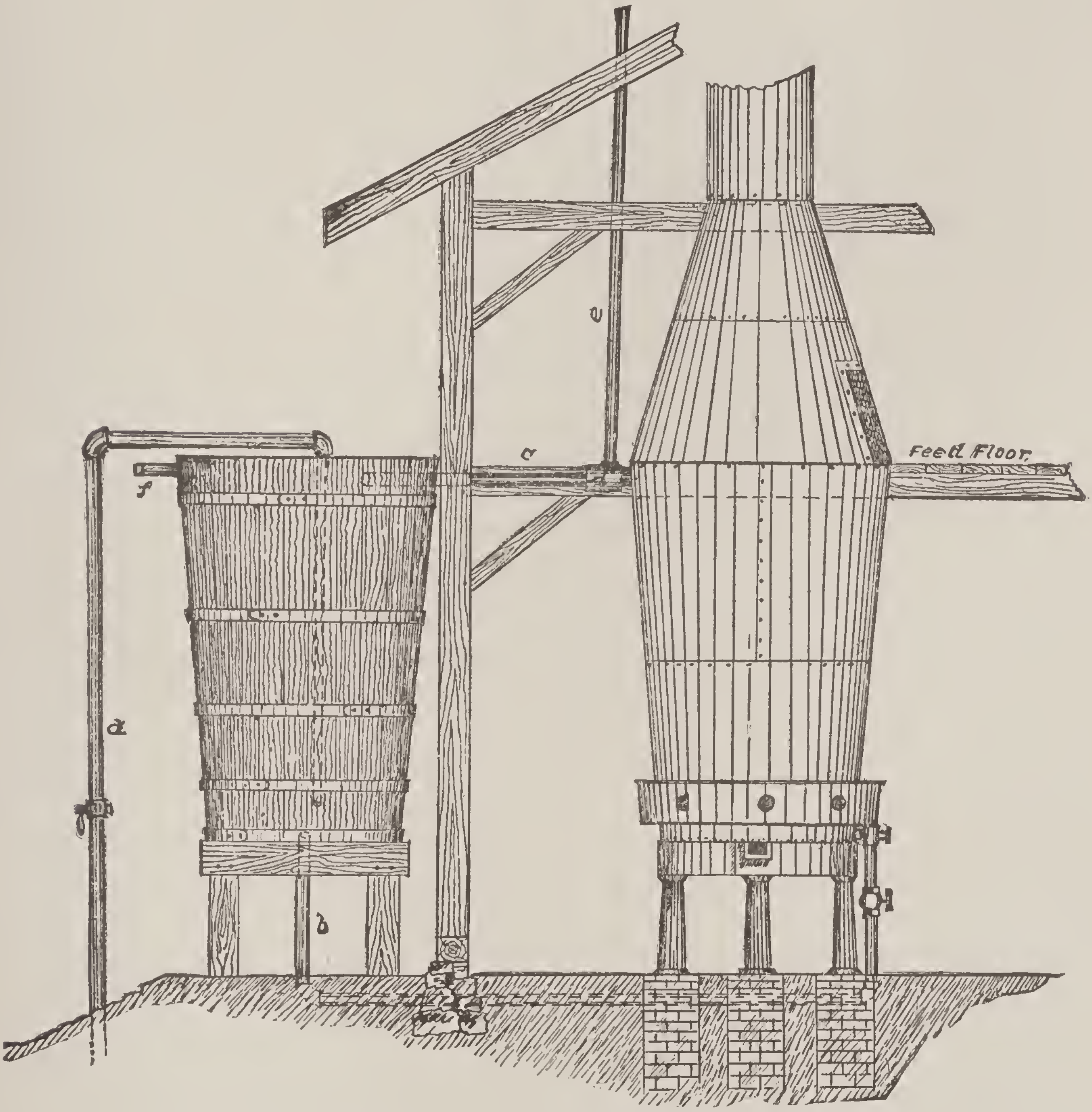
Water Jacket
Smelting Furnace
with
Carl Henrich's
Improvements
for
Copper Ores.

Manufactured by
Fraser & Chalmers
Chicago, Ill.,
U. S. A.





ject, and any reliable details of their performance, capacity under differing conditions of blast and charge, etc., are valuable. On this account, the table given above, compiled from the records of practice, although not as full or complete as desirable, will still be found of value.



BARTLETT WATER-JACKET—WATER-TANK ARRANGEMENT.

CHAPTER X.

BLAST-FURNACES CONSTRUCTED OF BRICK.

THE small type of brick cupolas, though not yet entirely abandoned in the United States, in the treatment of copper ores and mattes, has been described in almost every former work on this subject, and it possesses no peculiar or distinctive features that demand particular attention.

The largest type of brick cupola as yet found unmistakably practicable and advantageous will be selected for detailed description, the accumulated experience of several years, and covering almost every grade and variety of copper-bearing material, having emphatically demonstrated its economy and general superiority.

That the bounds of economy have not been overstepped in this matter of size is evident from careful comparative experiments, which show conclusively that the cost per ton of ore increases, the repairs become proportionately greater, and the ease of management is sacrificed with every inch that is taken from the size already referred to. What the limits may be in the other direction is yet an open question; but experience has shown that any further considerable augmentation of capacity involves the solution of various new problems pertaining to the blast and to the handling of such large quantities of ore and slag, and certain other matters that will be noticed in their proper place. It would be unjust to attempt the history or description of the successful introduction of this form of large rectangular brick furnaces without mentioning the names of certain persons whose perseverance and skill have overcome the difficulties inseparable from such an undertaking, and who have made to American metallurgy one of its most valuable additions.*

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* The gentlemen referred to, Messrs. W. E. C. Eustis, R. M. Thompson, H. M. Howe, J. L. Thomson, are, or have been, all officers of the present

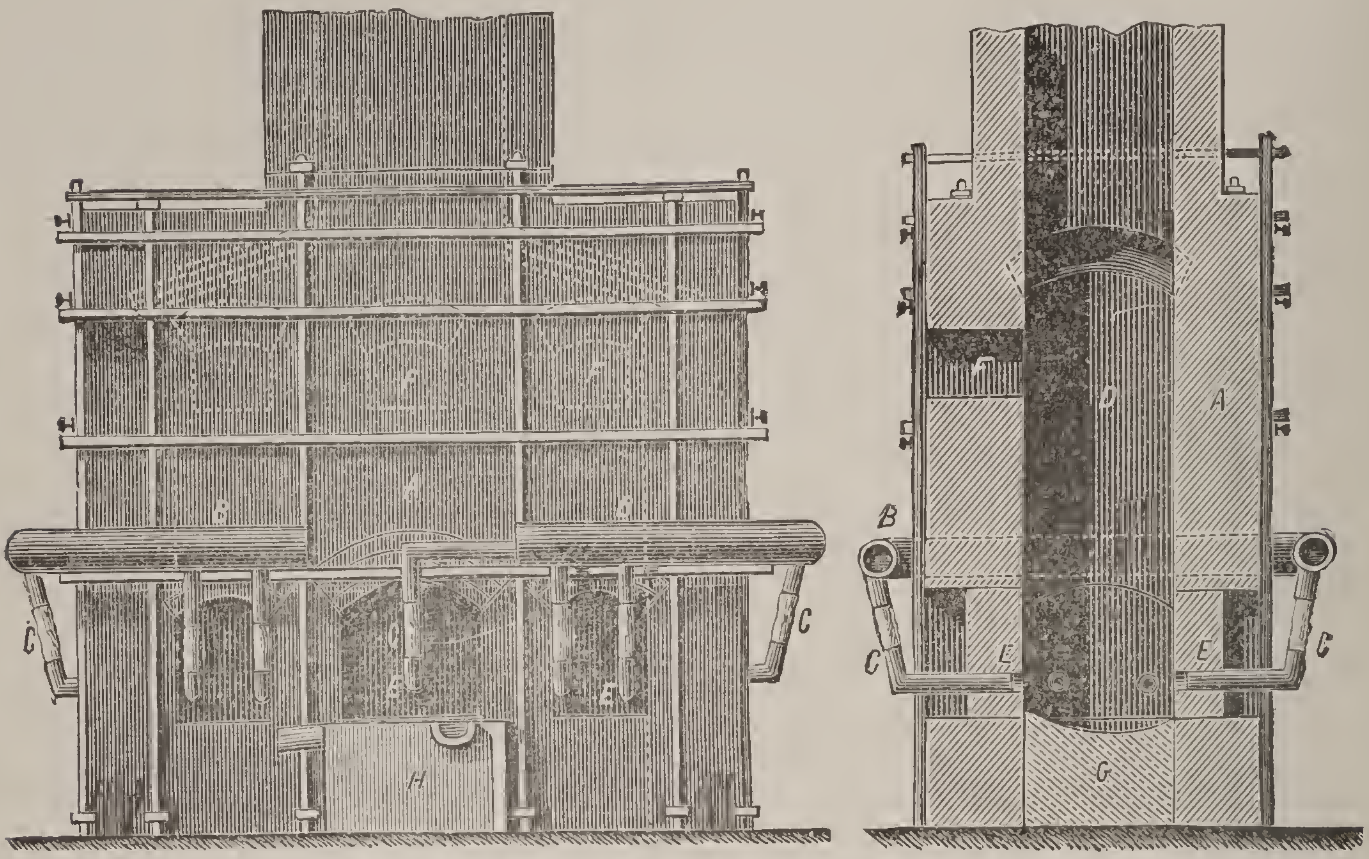
The distinctive peculiarities of the "Orford" furnace, as this altered and improved form of Raschette furnace is usually designated, aside from its unusual size, are the large number and diameter of its tuyere openings—14 of 6 inches diameter; the absence of any interior crucible or space for the collection of the fused products; the substitution therefor of an exterior forehearth or basin, and the construction of the latter in such a manner that two continuous streams—of slag and metal respectively—flow therefrom into ordinary slag-pots, without any blowing through of the blast, or delay for tapping and other related manipulations. The latter arrangement may be applied to any furnace of sufficient size, it being absolutely essential, for the prevention of chilling, that a large quantity of molten material should constantly traverse it. If the product is a matte of high grade, 60 per cent. and over, a much larger quantity is necessary to prevent chilling than if the metal is of poorer quality. The rapid chilling of the former is due not to its possessing a higher fusion point, but because its capacity as a conductor of heat increases with its percentage of copper.

When the smelting mixture is exceedingly rich, so that a very large amount of the copper-bearing product results, it is even possible, by rapid smelting, to maintain a constant stream of metallic copper—a practice that may be regarded as a curiosity rather than as ordinarily feasible.

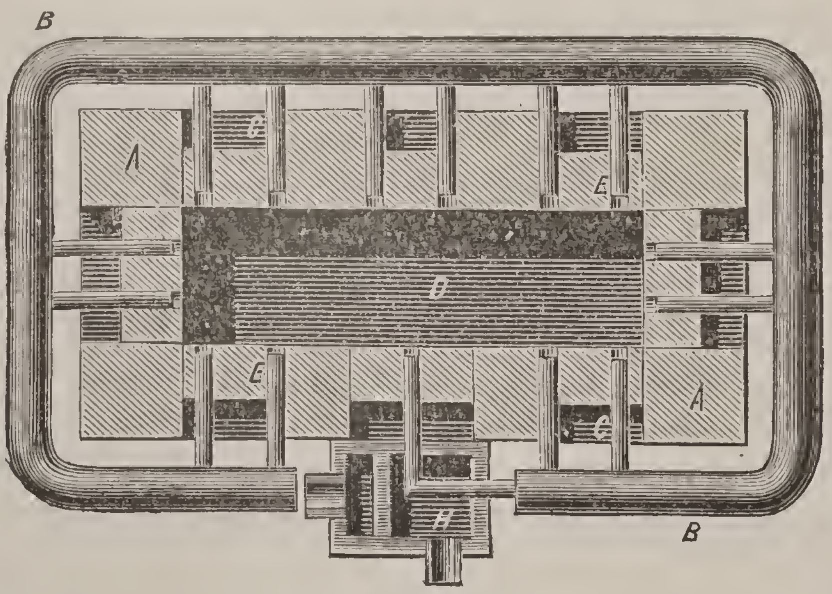
Orford Copper and Sulphur Company, of Bergenport, New Jersey; and while the furnace referred to closely resembles the *Raschette* type of furnace so extensively introduced into Germany and Russia during the past thirty years, its management and all manipulations connected therewith are sufficiently different to convert into a brilliant success what has in Europe, at least, been practically a failure. The application of the exterior crucible, continuous matte-tap, and peculiar method of feeding and manipulation by which campaigns of a year are made with an excessively basic slag, and in the entire absence of any water-cooled tuyeres, aside from the trebling of its former capacity, are sufficient to constitute a valid claim to originality.

The writer is pleased to have this opportunity to acknowledge the benefits derived from long and intimate intercourse with these gentlemen, and to state that it was while occupying the position of superintendent of this company that he first learned the full extent to which the cost of smelting could be reduced by increase in the capacity of furnaces.

A detailed description of the construction and subsequent management of this form of furnace will bring forward the points already referred to, and illustrate the practice that up to the present time has been found most advantageous, and which has cheapened the smelting of copper ores to a remarkable extent.



ORFORD BRICK FURNACE.



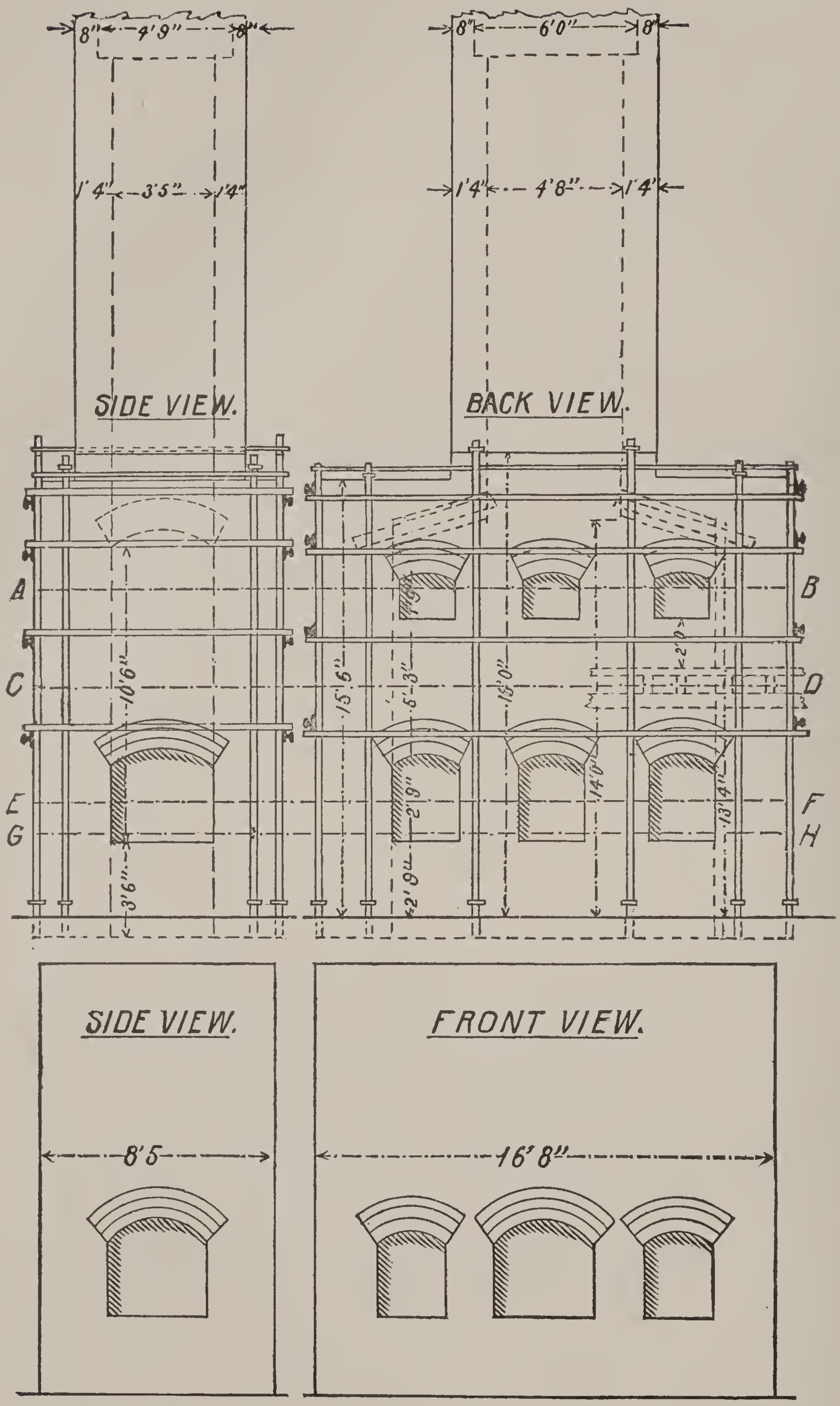
PLAN.

The outside measurement of the furnace being 8 feet 5 inches by 16 feet 8 inches, an excavation should be made at the intended site some three feet larger in every direction than the figures just given, and of sufficient depth to reach solid ground and insure a proper foundation. A depth of 4 or 5 feet will usually suffice, the pit being immediately filled with concrete; or, where possible, the pit should be filled to nearly the surface with molten slag.

The walls of the furnace should be begun a foot below the ground level, and should consist entirely of fire-brick up to the tuyere level, where the panels, shown in the cut, are begun. Up to this point, the walls are 30 inches thick, of solid fire-brick, while the panels are only 18 inches thick, thus being more accessible for repairs, and containing the tuyere openings. The rear wall is divided into three panels, equally spaced, and supported on each side by the full thickness of the wall, forming columns at each corner, and between the weaker portions, that are chiefly relied upon to carry the weight of the superincumbent structure. The panels are 30 inches wide and 33 inches high, and are strongly arched over with three rows of fire-brick, above which the full thickness of the wall (30 inches) is maintained to the top of the structure. Each panel is pierced by two 6-inch square tuyere-holes, equally spaced, excepting the central front panel, which contains only a small orifice for the slag-run, at a point some 10 inches below the tuyere level. The panel referred to forms the breast of the furnace, and is not closed in until the last moment.

The total number of tuyere openings is 14—6 behind, 4 in front, and 2 at each end. The interior rectangle is 3 feet 5 inches wide and 11 feet 8 inches long, although any exact adherence to these measurements is unnecessary, the interior of the furnace being soon burnt out into an irregular shape and usually much larger than the size just given.

Strong tie-rods, provided at their extremities with loops, and buried deeply in the foundation, are placed in position as indicated in the cut. Unless the transverse rods can be placed at a depth of two or three feet below the surface, they should merely be fastened into the wall by hooks, as they would certainly be melted away in time.

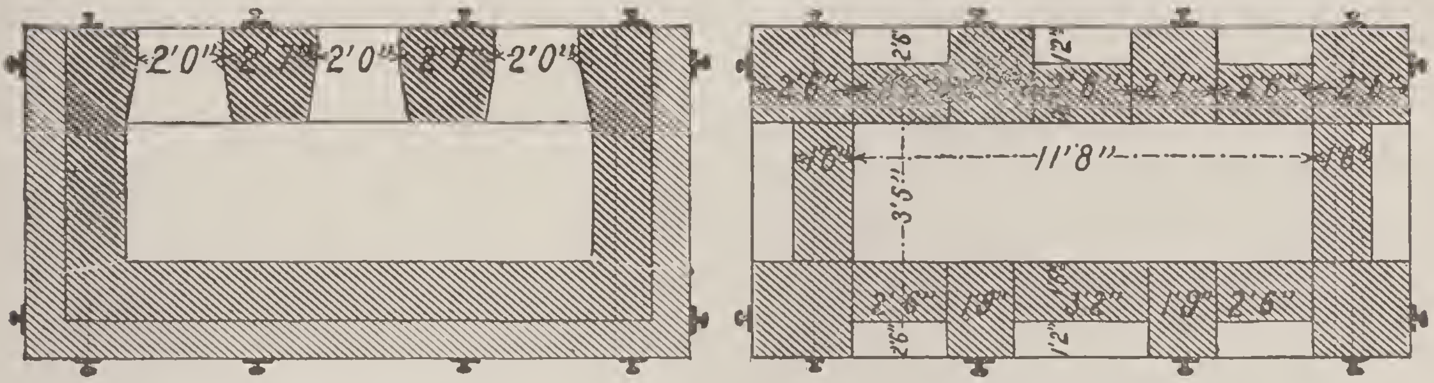


SCALE 1/8 IN. TO THE FOOT

The brick should be laid with the closest possible joints, and in a very thin mortar made of half each of raw and burned fire-clay, ground exceedingly fine.

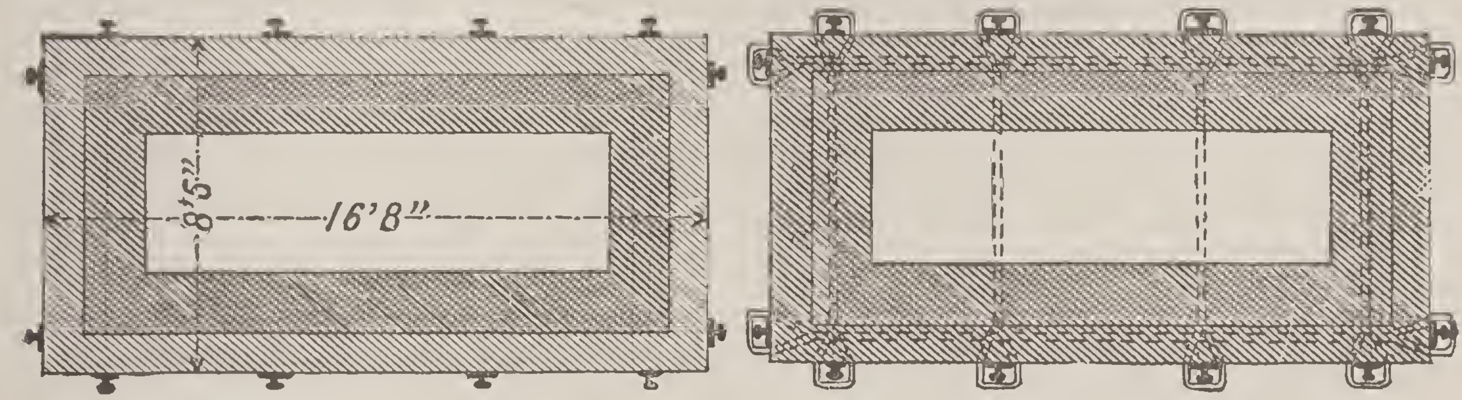
Heavy railroad iron may be used for binders, and should be used rather more than less liberally than shown in the illustration, as the expansive force is enormous when the furnace is in full heat, and any serious cracking tends greatly to shorten its existence.

If fire-brick are expensive, the outside lining, above the panels, and to a depth of 12 inches, may be constructed of red brick, although this is not recommended.



SECTION A B.

SECTION E F.



SECTION C D.

SECTION G H.

THE ORFORD "RASCHETTE" FURNACE.

The usual height from the tuyeres to the threshold of the charging-door is 8 feet; but this, of course, may be varied to suit the character of the ore to be smelted. The charging-doors are three in number, and of large size. All further details of construction are plainly shown in the cut.

The chimney should never be made smaller than here shown, and if a vertical down-take is used, connected with flues for the saving of the flue-dust, its dimensions should be increased one-third. The latter construction is much preferable to the simple vertical chimney, and is absolutely essential where anything but the poorest material is smelted, as the loss in flue-dust, owing to the enormous volume of blast peculiar to this practice, is very great—especially as a large proportion of the charge often consists of fine ore, it having been found that these large rectangular furnaces are peculiarly adapted to the treatment of that material.

The tuyeres consist of rather heavy, galvanized sheet-iron—No. 18—and are connected with the vertical branches of the main blast-pipe surrounding the furnace with thick duck tuyere-bags, soaked in a strong solution of alum, to render them less inflammable and to fill the pores of the cloth. Their diameter may vary with the character of the ore under treatment, but is usually from five to six inches, the pipes being merely thrust a short distance into the square orifices left in the brick-work, and made tight with plastic clay.

There remains nothing in the construction of this furnace that cannot be plainly seen from the illustration, and the discussion of its management from the time when taken in hand by the smelter will now be proceeded with.

It is frequently customary to form the bottom of a solid mass of fire-brick, placed on end, and brought up to within 10 inches of the tuyere openings, sloping slightly toward the slag-run in the center of the front wall.*

The author has found the following method, practiced originally by the Orford Company, far superior to any other, especially where low-grade matte is to be produced, the most difficult of all copper-bearing materials to confine within brick walls.

After filling in the foundation with béton to a foot below the ground level, the furnace bottom is begun by laying two courses of fire-brick on end, and with the closest possible joints. This still leaves a space of from 24 inches to 30 inches to bring the bottom to the proper height, which is filled in as follows:

The furnace and foundation being thoroughly dried by at least four days' brisk firing with brands and similar material, enough coke is dumped into the red-hot shaft to fill it to a point some three feet above certain temporary openings that should be left in the brick-work while building. These openings correspond in size, number, and position with the permanent tuyere openings, except that they are some 8 inches lower and directly beneath the regular orifices, which, for the present, are plugged with clay.

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* The practice of basing the bottom upon an arch built over an open space below must be strongly condemned, as it will simply result in the cutting through of the arch, and the total disappearance of all metal until the cavity is filled, making eventually a solid, but somewhat expensive, bottom.

Some six or eight tons of calcined quartz crushed to the size of chestnuts and mixed with about 5 per cent. of fusible slag, are spread upon the coke; and as soon as the latter is properly on fire above the temporary tuyere openings, the blast-pipes are put in place, and a light blast is continued until the coke is burned away, and the sticky, half-melted charge threatens to flow into the tuyere openings. The unconsumed coke and excess of quartz are removed through the breast panel—which was built up temporarily of 4-inch brick-work; and the furnace, being tightly closed, is allowed to cool very gradually for twenty-four hours or more.

If the operation is successful, the bottom will be as solid and infusible as can be made, nor will any attempt at the substitution of basic material for quartz, in consideration of the probably highly ferruginous character of the slag to be produced, result in any improvement on the plan recommended.

It is probably as good a bottom as can be made, although, as will be later seen, it offers but little resistance to a hot low-grade matte, when produced at the rate of from 30 to 50 tons daily.

The furnace being thoroughly dried and heated, blowing in may follow at once, it being only necessary to plug the temporary tuyere orifices, fill the shaft with coke to a point some 3 feet above the permanent tuyeres, and allow the fire to ascend to these openings before filling the shaft with alternate layers of charge and fuel, and putting on a light blast (one ounce).

All this may be done the night before starting, and at the same time, if not before, the fore-hearth and siphon-tap* must be arranged. This consists of a rectangular box, some 4 feet by 3 feet 6 inches, formed of cast-iron plates strongly bolted together at the corners, and lined with a brick wall 4½ inches or 9 inches thick, according to the quality of the product. It is fastened firmly to the front of the furnace, just at the slag-run in the center panel, the lower middle portion of the anterior front wall of that structure forming its posterior boundary. It is divided longitudinally by a 9-inch wall of fire-brick into a greater and lesser portion, the area of the two compartments being about as 5 to 2, and the direction of the division wall being parallel to the short axis of the furnace.

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* This is an entire misnomer, as the apparatus here referred to, as used for the continuous discharge of the metallic product, has nothing about it pertaining to the principles of the siphon.

The entire molten contents of the furnace discharge through a 2-inch by 4-inch opening (the slag-run) in the middle panel (the breast) into the larger of these two compartments, which is provided with a slag-spout, bolted to the upper edge of the front plate, while it communicates with the smaller compartment by means of a 3-inch by 8-inch vertical slot through the 9-inch division wall, about midway of its length and on a level with the floor of the fore-hearth. This smaller compartment also has a spout about 2 inches below the level of the spout belonging to the larger division, and on the outer side, instead of the end wall, for the sake of convenience.

A thorough understanding of this very simple and inexpensive contrivance will render it very easy to appreciate its management.

When the breast-hole is opened, and slag and metal first begin to flow, the larger compartment is soon filled, as the only means of communication between the two divisions of the fore-hearth is the closed slot in the lower part of the 9-inch division wall.

The molten products separate according to the law of gravity, and slag is allowed to flow through the spout of the large compartment until the drops of metal appearing show that it is filled with the more valuable product. The channel of communication is now opened by means of a crooked tapping-bar, and the metal flows rapidly through the same into the smaller compartment, until an equilibrium is established, and both divisions of the fore-hearth are partially filled with the matte, the communicating channel being far below the surface of the same, and consequently so situated that slag can never reach it unless it should sink below the metal, which is obviously impossible.

As the furnace constantly discharges its stream into the larger compartment, the fore-hearth is soon filled again, the metal sinking to the bottom and standing at the same level in both divisions, while the slag simply flows over the surface of the matte in the larger compartment.

As soon as the matte reaches the level of the spout attached to the small compartment, it begins to flow into a pot placed to receive it, and by judicious manipulation, and if a sufficient proportion of matte is produced from the charge, a constant stream of each product may be kept running without difficulty.

The management of this "siphon-tap" requires considerable experience, as the matte stops occasionally without apparent cause, and requires a certain amount of manipulation and coaxing to keep running freely. This is accomplished by slightly damming up the slag-spout, which soon forces an excess of matte into the smaller compartment, or by clearing out the communicating orifice by means of a heated bar bent to the required curve.

With matte of 50 per cent. or over, the principal difficulty is found in the gradual filling up of the fore-hearth by chilling, while a matte containing 20 per cent. or less of copper, and produced in large quantities, has directly the opposite effects, thinning the fire-lining until the plates are endangered, and cutting away the division wall until the two compartments are virtually thrown into one.

But even under these circumstances, and as long as a vestige of the center wall remains, the separation of the matte and slag continues to be perfect, and by judicious repairing and nursing, a fore-hearth apparently in the last stage of ruin may yet do good service for many days.

An opening through the division wall 18 inches high by 24 inches wide, and actually involving two-thirds of the separating brick-work, is not incompatible with a perfect separation.

The larger compartment is provided with a tap-hole at its lowest boundary, and on the side opposite the matte division, and a large quantity of sand should always be at hand ready to make up into rough molds in case of any sudden necessity for tapping the furnace.

This is especially the case when producing very low-grade metal; for owing to its corrosive action, and to the fact that the anterior wall of the furnace forms the posterior boundary of the fore-hearth, the entire contents of the former may escape into the latter in case of a break through the plates. It is at first somewhat startling to have such an outbreak when the entire bottom of the enlarged and burned-out furnace has been excavated to the floor level, forming a crucible some three feet deep and perhaps 4 by 13 feet in size. Under such circumstances the emptying of the fore-hearth by tapping—or oftener by breaking through the plates or brick-work at some point—may result in the irruption of some 12 to 15 tons of matte upon the floor of the cupola-house. The workmen soon become expert at controlling such outbreaks by means of dry sand in unlimited quantities—the approach of anything wet is like touching a match to a keg of gunpowder—and no serious results need be apprehended when the buildings are fire-proof, as should invariably be the case where large brick furnaces are employed.

Such an outbreak is treated as are most other accidents to which this type of furnace is liable, by entirely shutting off the blast and allowing everything to stand quiet for a few hours. The orifice is tightly plugged from the outside, and the molten products that trickle into it from the interior are allowed to cool by standing still, until it is as tight as ever.

The full burden may be reached after feeding two quarter charges, four half charges, and eight three quarter charges, slag being substituted for ore to a considerable extent, until the condition of the furnace warrants the employment of the normal mixture.

This is shown by the gradual change of the color of the slag from a dull red to a yellowish white; the entire ceasing or great diminution of smoke arising from the slag; a certain peculiar viscosity (except in very basic slags) when it falls into the pot; a general brightening of the tuyeres, succeeded by the formation of short noses, perforated abundantly with bright holes; and a steady and rapid sinking of the charge.

Although the charging of the blast-furnace is always one of the most important manipulations belonging to this apparatus, it is doubly the case with the furnaces now under discussion.

While the walls of the water-jacket are thoroughly protected and entirely unassailable, the mason-work of the brick furnace is completely exposed, and any error in the proportion of fuel to ore or in the manner of charging is sure to be followed by serious results.

This is, strange as it may seem, peculiarly the case with a siliceous charge, and nothing can more clearly illustrate the proper method of working than a brief description of an irregularity that is constantly liable to occur, and that will be quickly recognized by all practical cupola smelters.

An imaginary case will be assumed where a newly blown-in furnace, in good condition, but with a slightly too siliceous charge, begins to become too hot in one end, through some slight irregularity of feeding, or through an improper proportion of ore to fuel—either too much or too little of the same producing very similar effects.

The attention of the foreman will be called to the fact that one of the end panels is becoming very hot, which, as it consists of 18 inches of fire-brick, shows either that the inner temperature is much too high, or that the bricks have already been thinned by burning.

A glance into the tuyere opening shows that a heavy black nose has already formed, resulting from the fusion of the fire-brick above, which form a crust almost impervious to a steel bar, and exceedingly infusible.

A consultation with the man who feeds that end of the furnace will elicit the information that that portion of the charge is sinking very slowly, and that the heat is rising to the surface.

At the same time, the blast-gauge will show an increased tension, owing to the blocking up of the tuyeres that supply that portion of the apparatus, and the agglomeration of the charge above, owing to the rapidly ascending temperature.

The already too siliceous slag is rendered still more infusible by the admixture of silicate of alumina from the melting fire-brick; and the high temperature and powerful reducing atmosphere, resulting from the almost stationary condition of this portion of the charge, soon begin to reduce metallic iron out of the slag, and even from the matte, the sulphur being driven away to a considerable extent by the powerful blast, high temperature, and slow removal of the molten products.

The slimy, half-fused metallic iron is soon recognized by the bar which is constantly thrust into the choked tuyeres, and the inexperienced metallurgist, following the teaching of all our best text-books, reasons that the reduction of iron comes from too highly ferruginous a charge, and destroys all hope of improvement by cutting off a portion of the iron from the charge fed into that end of the furnace.

This further diminution of the oxide of iron, and consequent necessary increase of temperature to melt the more and more infusible slag, soon bring about the exact conditions prevailing in an iron-ore blast-furnace. Metallic iron is reduced in large quantities, while the temperature is raised several hundred degrees, before the slag—now virtually an acid silicate of alumina and lime—will become sufficiently softened to run at all. In the meantime, the furnace wall, at the panel, is burned nearly through; jets of blue flame appear at every joint and crevice, and the most superficial examination shows that the process is extending into one or the other of the corner columns, threatening the stability of the structure, and still more alarming the person in charge. The column of ore in that end of the furnace hardly sinks at all; the heat is ascending to the surface of the charge; and the general increased stickiness of the rapidly lessening slag-stream, increase in tenor of the matte, and deposition of lumps of metallic iron in one or both compartments of the fore-hearth, show that the end is not far off, and unfold the near prospect of a chilled furnace, and the probable presence of a block of half-molten ore and iron that is almost impervious to tools, and may result in the entire abandonment and destruction of the furnace.

This is one of the most common and well-known occurrences in small furnaces and with inexperienced metallurgists, and might just as well happen to the large furnaces now under discussion, were it not fortunately that their construction and management are not likely to be undertaken except by men of experience, and also that, owing to their greater size, a threatening—or even established—chill is much more easily managed than in the case of the smaller cupolas, whose contracted shaft is filled up solid almost before one is aware that anything is going wrong.

Owing to the great area of the Orford furnace, a considerable portion of the shaft may be completely blocked by a chill, while a brisk fusion is progressing in the other half, giving an opportunity, by the use of skill and experience, to gradually smelt away the solidified portion and eventually bring matters back to their normal condition.

Returning to the imaginary case that has just been followed to a disastrous termination, the writer will endeavor to show how such a catastrophe may be averted, and will describe the course of events as they have occurred scores of times to every practical smelter.

The moment that it is noticed that one end or corner of the furnace is becoming abnormally hot, and that the column of ore corresponding thereto is sinking slowly, the tuyeres belonging to that portion of the shaft—from one to three in number—are immediately removed, and the openings slightly plugged with clay. At the same time, several charges of the most fusible slag—that from matte concentration and containing a very high percentage of iron is best—are given, in place of ore, and the whole furnace is most carefully watched, to learn whether the burning is due merely to some local irregularity in feeding, or whether some important point affecting the whole process is at fault; such as too much or too little fuel in proportion to ore; improper composition of slag; incorrect feeding; too strong or too weak a blast, etc., etc.

Experience alone can qualify the metallurgist to quickly and correctly detect the cause of the trouble and apply the appropriate remedy; but in any case, if, after taking the precautions enumerated and waiting a sufficient time to get their full effect, the burning still continues, it becomes evident that the trouble is deep-seated and of some extent.

Vigorous measures are therefore required to stop the melting of the brick-work above the tuyeres, and not only to cool down the heated end of the furnace, but also to repair, as far as possible, the damage already done to the panels—or even to the corners of the main columns.

Still keeping the offending tuyeres closed as already described, a full charge of siliceous ore should be fed in such a way that it will sink to the indicated spot. This may be given either with or without coke, or may be followed by a second or third, or even a greater amount, as the circumstances indicate; proceeding with extreme caution, and allowing some two hours to intervene between charges.

The author has found it necessary to charge as much as 11 tons of almost pure silica—quartz with specks and veinlets of carbonates and oxides of copper—into one corner of an overheated furnace, and this entirely without coke, before the gradual cooling of the external walls, normal and even sinking of the charge, and lowering of the temperature at the charging-door, indicated that the mischief had ceased.

The office of this siliceous addition is not to render the slag in general more siliceous. This would only bring about the evils already indicated, and probably cause a heavy reduction of metallic iron. Its object is rather to produce, by the sudden arrival of such a body of cold, infusible material, such an overwhelming effect as completely to cool down that portion of the shaft, the silica itself softening somewhat and remaining for the most part in the corner of the furnace corresponding to the point over which it was charged. It attaches itself to the walls and bottom, and fills up the cavity caused by the fusion of the fire-brick, lowering the temperature at the same time to a considerable extent, but producing no marked effect on the general character of the slag.

When this operation is successful, as is usually the case, the thinned and heated brick-work is virtually restored, the deeply excavated bottom is filled up to the general level, and matters resume their normal condition, all irregular bunches and protuberances of the siliceous addition that may have adhered to the furnace walls becoming gradually melted away and smoothed down until the interior mason-work, if visible, would be seen to have almost assumed its original appearance.

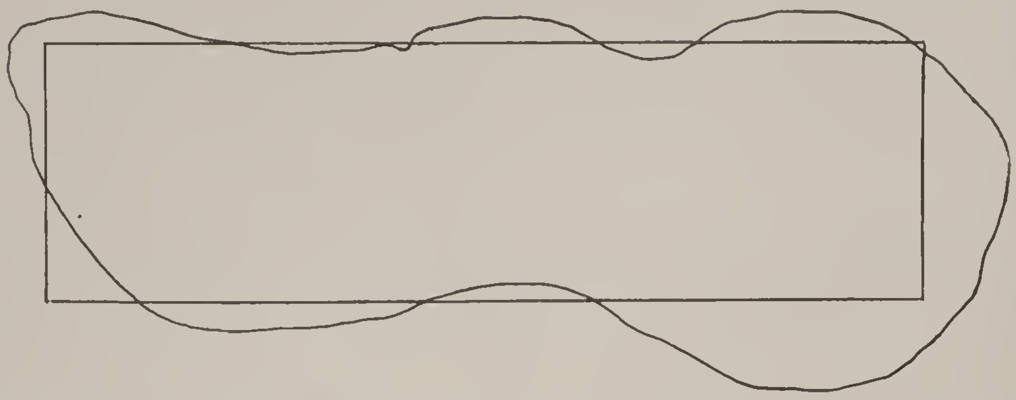
Such a result may seem very doubtful, and, in fact, the whole operation may appear to partake too much of the marvelous to those unfamiliar with such practice. The author would hesitate before describing the foregoing operation as a matter of general every-day occurrence, were it not that it can be vouched for in its entirety by a considerable number of well-known and reliable gentlemen. This practice, as initiated by certain members of the Orford Company, already mentioned, has spread until it is now a well-known and recognized part of our local copper metallurgy. The skill attained by certain foremen in managing these very large furnaces is quite remarkable, and far beyond anything described in this treatise.

While the imaginary case just described in detail represents only one of the various accidents peculiar to all forms of blast-furnace, it still is at the bottom of a very large proportion of the instances of "freezing," "choking-up," "burning-out," etc., etc. Paradoxical as it may appear, the two common accidents of "burning-out" and "freezing-up" are closely connected, and in reality only two different stages of the same morbid process. The young metallurgist cannot overestimate the importance of the fact that it is quartz in one or another of its forms that is the most frequent cause of smelting difficulties and disasters. Seven out of the last eight cases of metallurgical difficulties for which the writer was called upon to prescribe, were due to this cause.

In spite of the frequency and apparent simplicity of this difficulty, some smelters of experience never seem to have learned the cause, and attribute the slow and irregular running of the cupola and the frequent filling up of the crucible with sows to "too much iron in the charge"—"too much sulphur"—"magnesia in the limestone flux," etc., when, in almost every instance, a mere ocular examination of the slag is sufficient to show that silica is at the bottom of the trouble. No apology is needed for emphasizing this point, when men considered as expert metallurgists are constantly falling into this error.

It is especially during such accidents and irregularities that the great advantages of these very large furnaces become fully apparent. Where a small shaft would soon be completely and irretrievably choked, necessitating the great expense of blowing down and subsequently chiseling out the half-fused mass of ore and cinder, no large furnace, in any instance known to the author, has ever become so blocked up and filled with a chill that it has not been quite easy to save it by using appropriate means. Even though one end be completely blocked, there is always ample space at some points of its eleven-foot shaft to permit the descent of the charge and retain a sufficient number of tuyeres intact to gradually melt out the chill and restore the shaft to something like its former dimensions. Some considerable irregularity of form naturally results from repeated manipulations of this kind; but so long as sufficient area remains at the tuyere level, and no projecting masses impede the regular descent of the charge, no diminution of capacity need follow, nor increase of difficulty in managing the furnace.

The accompanying sketch gives a tolerably correct view of the shape of one of these large brick furnaces at the tuyeres upon its blowing out for repairs after a continuous campaign of 8½ months, during which time over 18,000 tons of exceedingly ferruginous ore were smelted in it, yielding a very low-grade matte and a slag averaging about 22 per cent. silica and over 70 per cent. protoxide of iron. As it is drawn to a scale, the extent of the irregularity is easily appreciable, the original dimensions being 3 feet 3 inches by 11 feet 4 inches.



THE RECTANGLE SHOWS THE SHAPE BEFORE THE CAMPAIGN; THE IRREGULAR LINE, AFTER THE CAMPAIGN.

In fact, the full capacity of this type of furnace, when smelting a basic ore, is not reached until the walls are burned out to a considerable extent, which may indicate the policy of widening the furnace in the first place. When smelting a siliceous ore, or when a large proportion of fines is present, the gain in width is accompanied with a decrease of temperature and irregularities in the descent of the charge—circumstances that soon rectify the trouble by adhering to the walls, and filling up the shaft again with a rapidity that may be disastrous if not observed and remedied in time.

As has been already briefly mentioned, the cutting down of the bottom and piercing of the foundation-walls is an accident that sometimes occurs, although usually only when the charge consists of a very fusible unroasted ore, producing a matte of low grade—from 25 per cent. downward—whose fiery and corrosive qualities are well known to all furnace-men. It is to the great quantity, as well as corrosive quality, of this substance, and this usually in connection with a basic slag, that this destructive process is due; and in spite of much care and expense bestowed on the matter, no material has yet been found that will withstand a daily production of from 20 to 45 tons of this intractable product. But a means of greatly lessening its destructive action, as well as of greatly prolonging the life of the entire structure and rendering its management much easier, has been discovered and quite generally adopted, being first brought into notice by Mr.

John Thomson, of the Orford Company. It consists in duplicating the furnace plant and running each individual cupola only ten or twelve hours of the twenty-four. This is a scheme that seldom recommends itself to one on first hearing, but, after a thorough trial, will be found to possess numerous important advantages, while its only drawback is the increased first cost of the plant—a trifling consideration in comparison with the large interests usually at stake.

A mere doubling of the cupola plant is sufficient to overcome the difficulties mentioned; but if it be desired to reap the full advantages of the scheme, a corresponding increase should be made in the blast apparatus. This being effected, the entire smelting process may be confined to the daytime, avoiding the difficulties and drawbacks of night work, saving the wages of one or more foremen, and rendering it possible for the manager to retain that complete personal oversight of the smelting process that is unattainable when half of it is concealed from his inspection. If this were the only benefit derived from the above plan, it would in most cases be well worthy of adoption; but the advantages accruing to the furnaces themselves, as well as to the entire process, are too numerous and far-reaching to be thoroughly explained in this treatise.

In the first place, the cutting down of the furnace bottom is usually completely remedied by the long and ever-recurring periods of complete repose, during which the thinned brick-work is again sealed by the chilling of the molten products; the hearth is renewed by the solidification of the matte and slag still remaining in the cavities of the hearth; the overheated brick-work cools from the outside to such an extent that the area that to-day has given constant annoyance by its obstinate burning, with the constant threat of finally breaking through and causing serious trouble, will to-morrow be found as cool as, or cooler than, any other portion, owing to the thinness of its walls; and various slight difficulties that are pretty sure to occur in the course of a long run are averted before they become of importance, while the trouble begins at a new point, only to be again averted before it has gained serious headway. This is by no means an uncommon or imaginary case, but a matter of frequent occurrence, and these lines are written after several years' trial of both the constant and intermittent method of smelting; the experience of others who have fairly tried this plan, in connection with large brick furnaces, being equally favorable.

The writer's attention was first called to this matter in 1871, when noticing the almost invariable improvement in behavior and capacity that succeeded any accidental stoppage of cupola-furnaces that he was then managing. The ores were exceedingly bad and siliceous, and the difficulties detailed in the preceding pages followed each other with disheartening regularity and frequency. Great pains were taken to secure a steady and uninterrupted run, fears being entertained that any stoppage would be disastrous to the furnace in the more or less critical condition that seemed to be its normal state; but after finding that the benefits following any temporary stoppage of the machinery had become so obvious that the foreman was in the habit of purposely causing slight accidents in order to help his furnace out of some particularly critical situation, it was decided to adopt the practice of stopping for two or three hours whenever the ordinary incidents of burning out, etc., became unusually critical. This habit was carried farther and farther, proceeding with caution and gradually lengthening the stoppages, until it came to be considered an almost universal remedy, and was as often applied for chilling or freezing up as for the opposite condition of affairs, and no misfortune ever arose from its reasonable application.

This practice, like every other, must be used with care and judgment, and may easily be carried to an extreme, but as a rule is the least dangerous measure that can be adopted with a badly acting furnace of large area. A small furnace might easily chill in a few hours, so that the length of the period of repose must be proportioned to the size of the shaft and to the cubic contents of the heated material. The thickness of the walls must also be considered, as the rapidity of the escape of heat depends upon the thickness of the brick-work. It is hardly necessary to say that every orifice and crevice about the furnace must be tightly sealed, the tuyeres being removed, and their openings, as well as the slag-run, being tightly filled with damp clay, while the brick-work in their vicinity must be searched for possible cracks, and all such openings carefully plastered over. Otherwise, the incoming currents of air would gradually burn away all the fuel contained in the charge, leaving the furnace in a hopeless condition.

If it is to stand still any length of time, such as over night, a few extra charges of coke should be given an hour or two before stopping, so that there may be an abundance of fuel in the bottom of the furnace. A small charge of basic slag should also be given; and as soon as the blast is taken off, the basin or fore-hearth tapped, and all openings sealed, the surface of the charge should be covered with a layer of fine coke, over which is spread an inch or two of fine fusible ore. The slag-hole connecting the furnace with the fore-hearth should be thoroughly cleared out; the layers of chilled slag and ashes, by which the blowing through of the blast is prevented, removed, and the channel itself filled with fine charcoal or coke, well rammed in with a "stopping pole." This is rendered impervious to air by an exterior plug of clay, and the fore-hearth, while still hot, being scraped clean of all half-fused masses of slag or reduced iron, and everything being prepared for the morrow's work, the cupola may be left in charge of an experienced watchman—preferably an old smelter. On the ensuing morning, a light blast is put on, and the channel being cleared out, slag will flow in from five to ten minutes, while in half an hour the furnace will be in normal condition, and in most cases smelting more rapidly and satisfactorily than when left the previous evening.

The extreme length of time that a large furnace may stand in this way without injury is unknown to the author. Much depends on the fusibility of the charge, the character of the fuel, the more or less perfect exclusion of all air, and probably also upon the quality and amount of sulphide compounds present, whose gradual oxidation may sustain the vitality of the charge for a much greater length of time than if absent. The following instances, from personal experience, show that a considerable delay is permissible.

A furnace running on a fusible charge of calcined pyritic ore was shut down Friday noon, on account of an accident to the engine. Further examination showed the accident to be of such a nature as to cause a delay until the succeeding Wednesday night —5¼ days—at the end of which time, a light blast was applied without much hope of a favorable result, although the coke on top of the charge was hot and glowing.

There seemed a good deal of obstruction to the blast at first; but in twenty minutes, a cold, thick slag began to run, which gradually improved, until the furnace resumed its normal condition and capacity in about eight hours. The charge had sunk about two feet in the furnace during this period of repose. The grade of the first tap of matte (the siphon-tap being impracticable in this condition of affairs) was 46 per cent., the ordinary average being from 28 to 29 per cent. The succeeding tappings gradually decreased—going successively 42, 37, and 34 per cent., the normal grade being reached soon after the furnace had regained its usual capacity.

Periods of 4 days, 3¾, 3½, 3, and of less time, appear in the writer's notes, the only serious accident, occurring during one of the shorter periods, being caused by the falling out of two of the tuyere-plugs, whereby a current of air entered the furnace for twelve hours before being discovered. The coke was completely burned out of the lower portion of the charge for about two-thirds of that part of the shaft nearest the opening; but the furnace was eventually saved by blowing lightly into three tuyeres at the opposite end, which were still supplied with fuel, and little by little smelting out the entire half-fused block of charge. Much benefit was derived by introducing coke into the furnace through such tuyeres as seemed to warrant the trouble. Owing to the great size of the tuyere openings (6 inches), this was easily effected, and the smelting much facilitated. In fact, if any cavity in the semi-fused mass could have been found at any point accessible to the blast, nothing would have been simpler than to break a hole through one of the brick panels and fill the opening with coke. The author has done this in later instances with very satisfactory results, a cavity opposite the tuyeres having been formed by dragging out a lot of the stock, from which the coke had burned so gradually as not to fuse it.

Space is wanting for a description of the use of petroleum, gas, and other concentrated fuels for similar purposes, as the writer's own experience with such measures has been entirely unsatisfactory, nor can he find any record of successful cases in the annals of American copper smelting.

The most Herculean efforts are warrantable when any reasonable probability exists of the saving of an iron furnace from complete chilling up; but in copper smelting, the comparative cheapness and simplicity of the structure itself, and the certainty of being able to remove the worst chill by mechanical means in a comparatively short time, render such unusual and expensive measures less important.

The oxidation of the sulphides in the charge during the period of repose is an element of some importance, although seldom so striking as in the case just mentioned. Still, the closing down of the cupola over night is invariably accompanied with a perceptible rise in the grade of the matte produced during a certain period succeeding; being greatest at first, and gradually diminishing as the contents of the furnace are replaced with fresh ore. This increase in richness is at first seldom less than 5 per cent., diminishing rapidly, however, as the ore nearest the bottom of the charge seems to have experienced the most thorough oxidation.

Though apparently a trivial matter, this enrichment of the matte is a direct pecuniary gain, and, according to a rough estimate, will offset the interest on the capital necessary for the double plant several times over in the course of a year.

Another useful and frequently applied remedy for various irregularities in cupola smelting is the so-called "running-down" of the furnace, by which is meant a mere cessation of charging until the column of ore and fuel has sunk to a point far below its normal limits. The shaft is then rapidly filled with the usual alternate charges of ore and fuel, and everything goes on as before.

Without attempting to explain the reason therefor, it is certain that this practice is sometimes of great advantage, obstinate irregularities often being conquered thereby, and the normal condition of things resumed. It is especially useful when it is desired to create a sudden and profound lowering of temperature at some point where a serious localized burning is taking place; for the exposure of the naked inclosing walls of the shaft renders it possible to deposit the batch of ore that is used to cool the walls in the exact spot where it is needed; and it is possible to use for this purpose, under such circumstances, an easily fusible ore or slag, instead of the highly siliceous material that is usually selected when this process of cooling down is undertaken blindly from above.

Wall accretions may also be reached in this manner, the charge being allowed to settle until they are exposed, whereupon they may be removed by a long bent steel bar introduced through one of the charging-doors, the glowing interior being cooled down, if necessary, by sprinkling with water.

Still another means of remedying the cutting-down of the furnace bottom has been mentioned in a former section, but is sometimes useful in connection with the large brick furnace. This is the introduction, through the tuyere openings, of ore or sand, which, being both cold and the latter infusible, will not combine with the slag, as it is already below the smelting zone; but will simply remain in place and assist in building up a new bottom. By this means, even the molten masses present may be partially solidified and a great advantage gained in a short time. The author has occasionally tried the introduction of water in the same manner and for the same purpose, taking as a guide the very decided local chilling produced by a leaky water-jacket; but the results, though locally satisfactory, are not sufficiently extended, while the operation itself, especially in connection with a low-grade copper matte, cannot be recommended to any who object to certain and frequent explosions of considerable force.

In connection with the measures already detailed for keeping the furnace in proper condition, may be mentioned the external repairs that it is feasible to execute while the furnace is still in blast. Not all smelters are aware of the very extensive repairs that may be carried out without stopping the blast more than a few hours; the length of the campaign often being doubled by the construction of a new panel, the repairing of a pillar, and other familiar and inexpensive operations. These are of too extensive and varied a nature to be enumerated in detail; but a few of the teachings of experience will throw some light on the practice in general.

The replacement of one or more panels that have become so thin as to threaten a constant breaking through of the charge is a simple, though very hot and laborious, task.

All needful material for the renewal being prepared and collected on one spot, the blast is shut off, the fore-hearth tapped, and the condemned brick-work at once broken in with sledge and bar. So much of the glowing charge as is necessary is at once dragged out of the opening with long hoes and rakes, and sprinkled with water so that the men can stand on it to work.

When the bricks have been removed to the extent deemed necessary, the cavity left in the column of stock is quickly filled with dampened coke, a few wooden slats being wedged across the opening, to keep the fuel from falling out.

The most important measure is to obtain a solid foundation for the new wall, and to accomplish this, all accretions of slag and metal of which the old wall largely consisted, must be chiseled away until sound brick-work is reached, which being leveled with thick fire-clay, offers a proper starting-point. The work must proceed with great rapidity, as the passage of air through the opening will soon consume the fuel in the charge. Little attention is paid to neatness, or even regularity, so long as strength and tightness are obtained. If the work promises to occupy more than two or three hours, the opening should be closed at the beginning by a thin plate of sheet-iron tightly cemented at the edges with clay, outside of which the new wall is raised. When all is completed, the sheet-iron—unless already consumed—is cut away opposite the tuyere openings, and the blast is put on at once, there being no necessity of waiting for the work to dry, as the heat from the furnace will evaporate all moisture quite as soon as is desirable.

By this means, extensive repairs may be executed on any portion of the furnace, it being even possible to put in a new bottom or repair the foundation walls, by suspending the charge on bars driven transversely through the furnace. When possible, the ashes of the rapidly consumed fuel should be cleared out before starting again; but there are but few instances where it will not be found better to blow out the furnace when such radical repairs are required.

The water-jacket furnace may also be allowed to stand idle much longer than is usually supposed, as the absence of air prevents the combustion of the fuel; but the rapid conduction of heat through its cold metallic walls prevents any such liberties as may be taken with the brick furnace, and renders it unsafe to leave the furnace more than twelve hours.*

In fact, it is better to run the charge down to the very bottom, throw in a few baskets of coke, and after stopping all the air-holes, leave the jacket in this condition, by which all danger of chilling is avoided, and the bottom being kept hot, smelting may be resumed in a very short time.

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* Since writing this paragraph, experience has taught me that water-jackets can be allowed to stand over night with as good results as in the case of brick furnaces, by employing the same precautions.

The final blowing out of the large furnace presents no peculiar features. The blast should be lessened as the charge sinks, and as soon as slag stops running, the breast-wall, and, if expensive repairs are imminent, some of the rear and end panels, should be knocked in, and all stock and fuel dragged out, until a tolerably even bottom is reached, which needs no preparation for the succeeding campaign.

Any burning out of the brick pillars that form the main support of this furnace, should be carefully watched and repaired before it has proceeded to a dangerous extent. This burning is sometimes so obstinate that when it is important not to stop the furnace or blow out, it is necessary to support the superincumbent brick-work with props and braces, which should remain in place until the pillars have been restored to their former strength.

Estimates of the cost of both building and running one of these large brick furnaces of the Orford type will be found in this chapter.

Similar estimates for the small brick furnaces formerly used at Ely, Ducktown, Ore Knob, and elsewhere, do not come within the scope of this work, which treats of *modern* rather than of historical methods. The cost of smelting in the small furnaces was from three to six times as great as in those now in use.

There remains to be still considered the application of water tuyeres and other cooling devices to furnaces constructed of brick or stone.

The author's own experience is entirely in favor of the employment of properly constructed iron, or better, bronze or copper tuyeres, containing a space for the introduction of water. In Colorado and other places, he has used water tuyeres with invariable satisfaction, the only drawback being the frequent cracking of the cast-iron, which is now overcome.

While they offer little or no protection to the furnace wall, they are indestructible themselves, and by delivering the wind at a fixed point, even though the walls may be eaten away all about them to the depth of a foot or more, they remove the point of greatest heat from the wall itself and practically retain the smelting area at the same invariable size, the latter being practically bounded by vertical planes passing through the nozzles of the tuyeres.

It is also possible, if desirable, to project them into the interior of the furnace to a distance of several inches from the walls. Although this practically diminishes the size of the smelting area, it saves the walls from burning, and in case of a weak blast or of an unusually dense charge arising from a large proportion of fine ore, may render practicable the smelting of material that would be impossible under other circumstances.

They were tried on the first large Orford furnaces, but failed, owing to the severity of the winter and other accidental causes, rather than from any fault due to the tuyeres themselves. Their construction and management are too familiar to require further explanation in these pages.

The surface cooling of the brick-work by means of a spray of water on the outside has been tried on many occasions and with various forms of apparatus. It has rarely given satisfaction, and in the writer's opinion is as dangerous and worthless a device as can well be imagined.

To those familiar with the results of contact between water and molten matte, it is not necessary to bring up any further arguments to condemn a device that can only be accompanied by a constant wetting of everything in the vicinity of the furnace.

Besides, the idea itself is an extremely faulty one, as, owing to the non-conductivity of fire-brick, a wall less than a foot thick may continue melting on one side, while its other surface is constantly sprayed with cold water.

All devices of this kind, in which the water comes in contact with the free exterior surface of the furnace wall, are, in the author's opinion, worse than useless, and likely to be accompanied by most dangerous results.

ESTIMATE OF COST OF LARGE BRICK BLAST-FURNACE.

Excavation for foundation: 1,000 cubic feet at 8 cents		$80.00
Foundation of béton		65.00
	Cubic feet.	
Total fire-brick for furnace proper	1,640	
Lining for cross-flue and down-take	540	
Fore-hearth, etc.	45	
Total	2,225	
Carried forward		$145.00

Brought forward		$145.00
At 18 brick per cubic foot = 40,050 at $40 a thousand		1,602.00
Red brick for down-take and flue: 16,800 at $8		134.40
6½ tons fire-clay at $8		52.00
6 casks lime at $1.50		9.00
2 tons sand at $1.50		3.00
Old rails for binders: 180 yards at 80 pounds a yard = 14,400 pounds at ¾ cent		108.00
Tie-rods for furnace, flue, and down-take: 620	Pounds.	
feet of 1¼ iron = 2,480 pounds	2,480	
Loops, nuts, etc.	166	
Angle iron for down-take	172	
Wrought-iron rods, etc., about fore-hearth	66	
Total	2,884	
At 2 cents a pound		57.68
Castings:	Pounds.	
3 feed-door frames	792	
Damper and frame	455	
Plates for fore-hearth	560	
Slag and matte-spouts	80	
Plates for charging-floor	1,260	
Miscellaneous	420	
Total	3,567	
At 2½ cents a pound		89.17
Material and labor for arch patterns and other carpenter work		32.40
Labor:		
Mason, 88 days at $4		352.00
Ordinary labor, 102 days at $1.50		153.00
9½ days, smith and helper		47.50
Blast-pipe and tuyeres		136.00
Cloth for tuyere bags and labor		3.80
Superintendence		120.00
Miscellaneous		65.00
Grand total		$3,109.95
Tools essential to furnace, steel, and iron bars, shovels, rakes, hammers		$55.90
15 slag-pots at $13.50		202.50
4 iron barrows at $9		36.00
Manometer		2.50
Total		$296.90

The above estimate is exclusive of main blast-pipe, blower, motive power, hoist, and chimney or dust-chambers; the allowance for cross-flue and down-take being sufficient to cover cost of chimney in those exceptional cases where no provision is made for catching the immense amount of flue-dust generated in this method of smelting.

A compact and economical hoist and ample provision for a large charging-floor and generous bin room are essential to convenient and economical work.

ESTIMATE OF COSTS OF CUPOLA SMELTING.

Details of expense of running a 42-inch circular water-jacket cupola, smelting 56 tons of fusible ore per 24 hours.*

As it may be a matter of interest to many to compare the cost of copper smelting in Arizona, Montana, and other remote districts with the cheaper scale of prices assumed for our standard, this information is given in a second column. These figures refer to works situated near a line of railroad, and of large capacity, as the smelters at a distance from travel must frequently pay double or even treble the amount given for coke and other supplies, while the cost of running a single furnace is proportionately much greater.

PER TWENTY-FOUR HOURS.

Fuel and supplies.	East.	Arizona.
Eight tons coke	$40.00	$200.00
Fuel for blast and attendance	7.50	16.00
Clay and sand	.60	1.50
Five tons limestone (or other flux)	7.50	15.00
Cost of pumping water for jacket	4.80	11.50
Oil, lights, etc	3.50	9.00
Renewal of tools, pots, molds, etc	2.25	4.60
Repairs on furnace and machinery	2.00	4.10
Proportion of cost of blowing-in and out	.40	.85
Sinking fund to replace furnace, etc	1.65	2.95
Miscellaneous	4.00	11.00
	$74.20	$276.50

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* These estimates, both of construction and smelting, are taken from the results of actual work, not being drawn exclusively from any one establishment, but being the average results of several successful works representing advanced American practice.

It must not be forgotten that several of the heaviest items that go to make up the running expenses of all metallurgical establishments are necessarily omitted. These are the general expenses and salaries; extraordinary expenses arising from accidents; cost of experimental work, and similar matters, which may aggregate a very large amount.

Labor (per twenty-four hours).	East.	Arizona.
Six men on lower floor	$10.00	$15.00
Four men on charging-floor	7.00	13.00
Two foremen	5.00	10.00
Two laborers	3.00	6.00
Proportion of blacksmith work	1.25	3.50
Proportion of laboratory work	2.00	7.00
Proportion of superintendence	3.20	10.50
	$31.45	$65.00
Fuel and supplies	74.20	276.50
Total	$105.65	$341.50
Costing respectively per ton	$1.88½	$6.10

To which should be added 5 per cent. for resmelting foul slag and flue-dust, increasing the final cost to $1.98 and $6.40 a ton. Nothing is allowed for transporting ore to the furnace and many other items, which only obscure an estimate supposed to refer to the cost of running a furnace as part of a larger plant.

If the entire expense of the works were supported by a single smelting-furnace, the estimate would be so complicated and the cost of smelting so high as to create an entirely false impression. Such instances occur, however, though only financially successful under exceptionally favorable conditions and with abundant and high-grade ores.

The following estimate of the cost of smelting a fusible ore in the large brick furnace so often referred to is also based upon the same conditions, the furnace being supposed to be only a portion of a large plant, and only to be charged with its own share of the cost of power, superintendence, etc., etc.

The ore is supposed to be a low-grade, roasted pyrites, or some other equally fusible and self-fluxing material. It is assumed that the furnace makes campaigns of nine months, smelting daily 95 tons of ore.

ESTIMATE.

Fuel and supplies:	
12⅓ tons coke, at $5	$61.67
Four tons pea coal for blower, at $3.50	14.00
Sand and clay	2.45
Oil, lights, etc	4.40
Wear and repairs on slag and matte-pots	3.85
Wear and repairs on other tools	1.12
Carried forward	$87.49

Brought forward	$87.49
Daily slight repairs on furnace	2.60
Proportion of radical repairs at close of campaign (found by experience to be 3 cents a ton)	2.85
Wear on belting, blower, etc	1.25
Engine and boiler	1.45
Proportion of cost of blowing in and out	.72
Sinking fund	1.95
Miscellaneous	6.50
Labor (per twenty-four hours):	
Six men below at furnace	10.00
Four feeders	8.00
Six wheelers	9.00
Two metal men	4.00
Two laborers	3.00
One dump man	2.00
Two foremen	5.00
One engineer	2.50
Blacksmith work	2.10
Laboratory work	2.00
Superintendence	4.00
Total	$156.41
Or a cost per ton of	$1.64⅔
Adding 8 per cent. for resmelting slag and flue-dust, gives total cost per ton	$1.78

CHAPTER XI.

GENERAL REMARKS ON BLAST-FURNACE SMELTING.

The capacity of a blast-furnace is dependent upon many varying causes, and is to a considerable extent independent of shape or size, though its tuyere area is, of course, the most important factor in determining the amount of material that can be passed through it.

Next to the fusibility of the charge, the pressure and volume of the blast have the principal influence in determining this point, assuming always that the fuel used is of sufficient strength and density to permit the full pressure of wind that may be found most advantageous.

Nothing can be more striking than the change in the rate of smelting of a large cupola-furnace as the wind pressure is diminished or increased.

The author has taken occasion during the smelting of a fusible charge, and with the furnace in perfect condition, to ascertain the difference of capacity effected by changes in the strength of the blast.

As the influence of the change is almost instantaneous, it is easy to arrive at such figures with considerable accuracy, measuring the capacity by noting the number of pots of slag produced during periods of an hour each, and with varying wind pressure.

The following table shows the result of these experiments in a compact form, repeated sufficiently often under varying conditions to establish their comparative accuracy.

It should be mentioned that, in order to insure the accuracy of each observation independently of the condition of the furnace previous to the experiment, which might have been influenced by the preceding test, nearly all the trials were made at different times, but with the furnace as nearly at its normal state as possible, and running under its ordinary pressure of blast—about 10 ounces per square inch:

No. of test.	Blast pressure in oz per sq. in.	Production in tons per 24 hours.	Assay of slag in copper.	Condition of furnace at close of experiment.
1...	½	16¼	0·27	Very hot. All tuyeres bright.
2...	1	21	0·35	Very hot. All tuyeres bright.
3...	2	31½	0·30	Very hot. All tuyeres bright.
4..	3	44	0·31	Slag hot and smoking. Tuyeres bright.
5...	4	64	0·31	Slag hot and smoking. Tuyeres bright.
6...	6	86¼	0·51	Slag hot and smoking. Tuyeres bright.
7...	8	87½	0·40	Slag still hot, but not quite so strikingly so as with lower pressure. Tuyeres satisfactory, but beginning to form noses.
8...	9	91	0·42	
*9...	10	99½	0·42	
10...	12	113	....	
11...	13	111	...	Less hot. Decided noses.
12...	14	115	0·66	Much cooler. All tuyeres require opening.

These tests, although not entirely uniform in every respect, are still quite regular, and agree closely with many previous observations.

With a light blast, the capacity falls rapidly, though the temperature rises, and the reducing action becomes more powerful, as evinced by the reduction of the oxidized copper in the slag, and the almost invariable appearance of small masses of metallic iron in the fore-hearth.

With the highest available blast, 14 ounces per square inch, the production still increases, though only slightly above the normal capacity, but it is evident more wind is introduced than can be consumed by the fuel; a lowering of temperature occurs, as distinctly shown by the appearance of the slag; the reducing action is less powerful, as seen by the slag assays; and thick, hard noses are formed about each wind stream, which would soon obstruct the blast, and probably cause a general chilling of the furnace.

Thinking that some of these evils might be attributed to the *volume* rather than the *pressure* of the blast, the tuyere openings were decreased from 5½ to 3½ inches in diameter, reducing the capacity of the furnace about 10 per cent., but otherwise effecting no visible alteration in the phenomena described.

Judging from this series of tests, as well as from numerous former trials, when smelting both lead and copper ores of many different varieties in cupolas of various sizes and under very varying conditions, it seems advisable to limit the blast pressure to the point just indicated. In no single instance has anything more than a temporary increase of capacity accompanied a blast pressure above 12 ounces per square inch, and the rapid cooling of the furnace and formation of heavy and solid noses have soon brought the experiment to a termination.

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* Normal pressure and slag assay.

It seems, therefore, that a pressure of from 8 to 12 ounces, with a tuyere diameter of from 4 to 5½ inches, is best suited to the ordinary conditions of copper smelting.

The employment of soft-wood charcoal or other fragile fuel may make it necessary to diminish even this light pressure, while anthracite may demand a more powerful blast for its most economical use. Information is wanting regarding the use of anthracite, but it is doubtful whether any advantage would be gained by its employment, its powerful reducing qualities causing an almost certain formation of sows.

Of the employment of a heated blast for copper smelting, the author must plead almost complete ignorance. The few details that he can gather on this subject indicate that no advantages commensurate with the cost of plant have been obtained by its adoption, and the tendency of a hot blast to increase both temperature and reducing action is obvious.

The common claim urged by inventors and manufacturers of smelting-furnaces is, that their apparatus is capable of generating a temperature much higher than ordinary furnaces.

This shows an entirely mistaken notion of the process of smelting, where our constant endeavor is, to prevent the temperature rising much above the point necessary for the fusion of the earthy constituents into a liquid and homogeneous slag.

The method of charging is pretty nearly universal, and differs radically from the old practice, where the establishment and preservation of a nose seemed to be the chief aim and end of the smelter's labors.

Both ore and fuel are now pretty generally spread in horizontal layers over the whole area of the furnace, instead of throwing the coke toward the center, while the charge was carefully placed against the walls.

The introduction of water-jacketed cupolas and the very general adoption of the conical shape, whereby the gases escape with less velocity, and the ore is forced to descend in the neighborhood of the walls, have doubtless initiated this method of charging, which has been followed with advantage by those who prefer the brick furnace, and still adhere to vertical walls. This mode of feeding, however, should by no means be blindly adhered to, as nothing exerts a more powerful influence upon the running of the furnace or has a more important effect in keeping it in normal condition than skillful and judicious feeding.

In the brick furnace especially, the position of feeder is one of vital importance, and the experienced furnace foreman will spend a large proportion of his time on the charging platform.

This matter has been discussed and exemplified in the section on large brick furnaces, and is worthy of the most careful study and attention.

The *absolute* size of the charge to be used must vary according to local conditions.

The most important of these are the area and height of furnace; mechanical condition of ore; nature of fuel; and extent of reducing action desired.

Large and high furnaces naturally require heavier charges of ore and fuel; a charge made up almost entirely of coarse material may safely be fed in thicker layers than if composed principally of fine dirt, which opposes a powerful obstacle to the passage of the blast; a heavy, compact coke will bear a much weightier charge than light, fragile fuel, like soft-wood charcoal; and a more thorough mixing of ore and fuel, as effected by using small charges, will undoubtedly bring about a more powerful reducing effect than when the different strata are of sufficient depth to retain their relative position to a considerable depth.

While very numerous exceptions exist, the author prefers, in general, large charges to small ones, having found, as a rule, that the furnace runs more smoothly and regularly, and also that a slight saving in fuel is effected.

This observation will no doubt be challenged by many competent metallurgists, but is the result of too long experience to be disproved without actual trial.

In only one instance has the writer attempted to determine this point by actual experiment; but in the case referred to, the conditions of the trial were particularly favorable for a fair and impartial comparison.

The furnace was a 42-inch water-jacket, smelting a mixture of reverberatory copper slag and fine unroasted pyrites, with gas coke as a fuel. The foreman, who was a most skillful smelter, was directed during the entire experiment to give his attention to the consumption of fuel, using no more than was necessary to attain the best possible results. The change in the size of the charge was made without directing his attention particularly to it. He was thus left to discover any necessity for a change in the weight of fuel.

The experiment was begun with large charges—1,480 pounds of mixture—the relation of the fuel to the same being as 1 to 9·3. This was maintained for 72 hours, the furnace remaining in excellent condition, and averaging 57 tons per twenty-four hours.

The charge was then reduced to 740 pounds, just one-half of the original amount, and twenty-four hours were allowed to elapse, to permit matters to find their normal level under the new conditions.

Within six hours of the substitution of the smaller charge, black noses began to form on the tuyeres, and the rate of smelting became decidedly slower. Several empty charges—that is, fuel without ore—were given at intervals; but it became evident, from increasing irregularities, that the furnace was growing cold. A slight addition was made to the fuel charge, and after a considerable number of trials, the normal ratio of fuel to ore for the new conditions was established, and the steady run resumed. A three days' average was taken, as in the former case, and showed the best possible ratio between charge and fuel to be as 8·6 to 1.

The charge was again halved, being now reduced to 370 pounds, and the last-named proportion of fuel maintained until circumstances compelled a change.

In brief, another three days' observation showed a further reduction in the ratio of ore to fuel—7·82 to 1 being the best attainable results. It is also interesting to note that, although great pains were taken to secure the same conditions in every particular during the entire course of the experiment, the matte decreased in tenor with the decrease in the weight of the charge—the average assay reports for the three periods of three days each, beginning with the heaviest charge, being respectively 46·4, 44·5, and 42·1 per cent.—the amount of the same increasing with its poorness in a very nearly corresponding degree. The slag also (although this may have been a coincidence) showed lower proportions of copper, assaying for the three periods respectively

0·61, 0·47, and 0·41, which is a greater difference than can be accounted for by the lower grade of the matte, and which in all probability, in common with the latter material, depended upon the more powerful reducing effect, due to the use of thinner charges, and a consequently more perfect mingling of ore and fuel. The capacity fell from 57 tons, in the first instance, to 51 in the second, and down to 41·5 in the third.

The experience at several Arizona furnaces contradicts the above results, quite small charges having been found to answer best, although this may be due to the fact that much of the ore there is fine, while a powerful reducing action is necessary to produce a clean slag.

A proper charge for a 36-inch furnace is from 500 to 800 pounds; while a 42-inch shaft should receive from 1,200 to 1,600, and a 48-inch furnace, 1,800 pounds or more. The large elliptical slag-furnaces at the Lake Refining-Works are charged with about 2,600 pounds of ore and flux, experience having shown the advantage of deep layers in the furnace shaft.

As may be imagined, the large Orford furnaces take still heavier charges, from 3,000 to 4,000 pounds being the ordinary standard.

The shape of the furnace is largely a matter of individual preference, as may be seen by observing the almost equal number of skilled advocates for the round, rectangular, and elliptical form.

Beyond a certain limit, however, the rectangular form alone is used, owing to the feeble penetration of the light blast used in copper smelting.

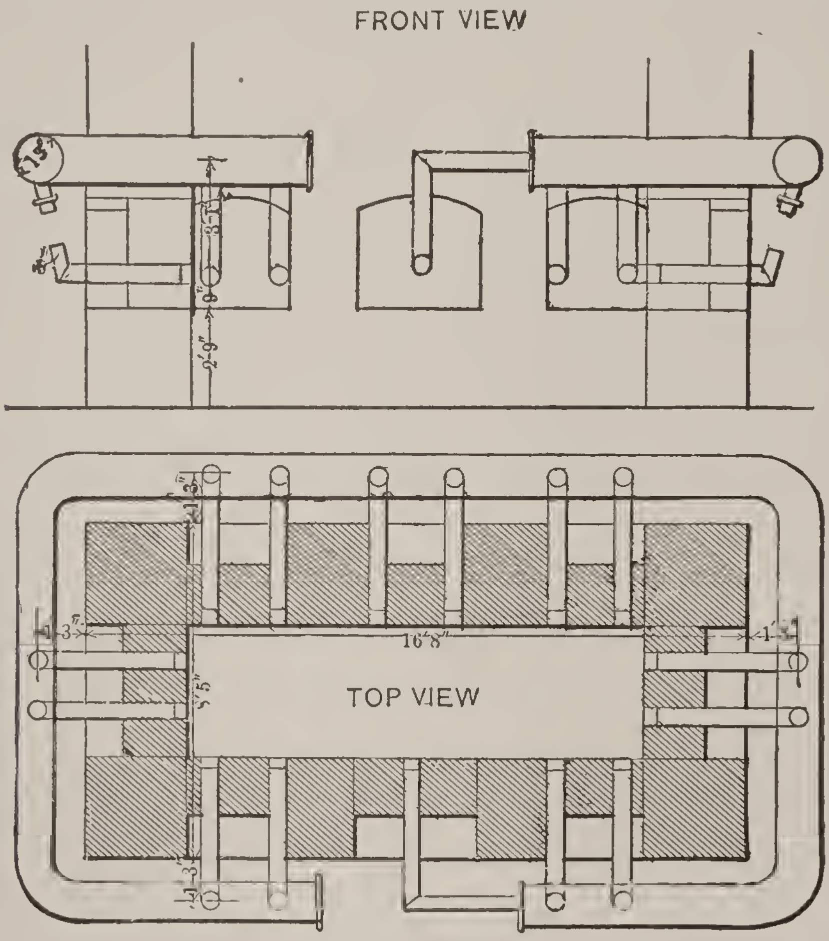
Experiments made by Herreshoff and other metallurgists, including the author, seem to indicate a radius of 28 inches as about the extreme practicable limit for a 10-ounce blast. In a larger furnace, while the writer has never seen any evidence of an untouched central core, beyond the penetration of the blast, the capacity increases very slowly, if at all; while the same area, when changed into a rectangular form, gives proportionately greater results.

If the charge contains over 50 per cent. of fine ore, the figures given above should be considerably reduced.

While the effects of a flaming throat are not so obviously detrimental in copper smelting as in the fusion of the more volatile metals, it still is found by experience that such a condition of affairs is incompatible with the best work, being invariably indicative of a faulty condition of the process.

With an open charge and long-continued high pressure of blast, it is almost impossible to prevent the heat from eventually rising, until the chimney and walls above the charging-door become so hot as to ignite the escaping gases instantaneously.

The ore near the top of the charge soon sinters together; the fuel is largely consumed before it reaches the zone of fusion; the softened lumps of ore stick to the side walls, forming bulky accretions, and the way is paved for the successive steps of "burning out," reduction of metallic iron, and "freezing up," already so frequently alluded to.



Scale ⅛ in to the foot

BLAST-PIPE FOR ORFORD FURNACE

While it is sometimes impossible to prevent the early stage of this condition of affairs, when pushing the furnace to its full capacity with a heavy blast, the end results should be borne in mind and the remedy applied in time.

This consists simply in letting the charge sink—under a light blast—until the shaft is empty for a distance of three or four feet below the charging-door. One or more charges of fusible slag are then given, and the furnace rapidly filled full with its normal burden. In this way, the overheated walls are cooled, the surface of the charge regains its normal temperature, and the furnace under a few hours of light blast is again ready for a period of hard driving.

In obstinate cases, the cooling of the throat with a spray of water is quite admissible, and often of great benefit.

The question of the characteristics and comparative value of the ordinary fuels used in blast-furnace work has been discussed so exhaustively in most of the standard works on metallurgy as to render it useless to undertake any such task in a treatise like the present, devoted to a certain stated purpose.

The same may be said of fire-brick and other refractory materials, our own domestic brick being quite equal to any of foreign make for all purposes connected with blast-furnace smelting. It is hardly necessary to say that, among the numerous competing varieties of fire-brick, only those should be selected which long and thorough trial has shown to be suited to the purpose; the first cost should have but slight weight in the choice.

The manipulation of the products of fusion becomes a question of considerable importance in a country where wages are high, and where the large scale on which most enterprises are conducted renders the mechanical details of the process so much more prominent than in the European works from which we drew our first patterns.

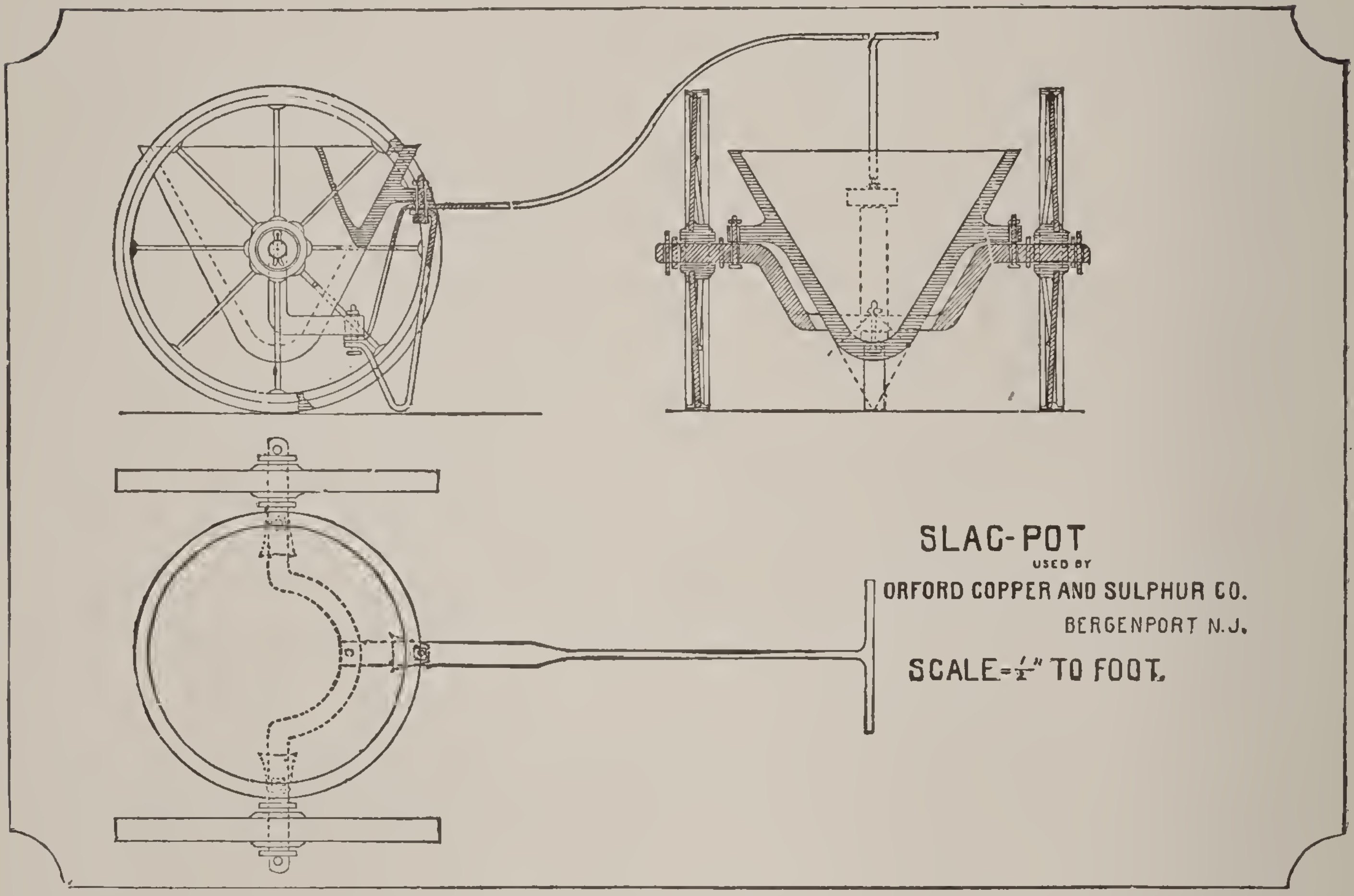
The transportation of the slag from a furnace smelting five tons an hour, and where the edge of the dump advances at a rate of several feet a week, soon becomes a matter of considerable expense.

Aside from the removal by a current of water, mentioned in an earlier chapter, no advance has been made on the two-wheeled slag-pot, although so much difficulty has at times been experienced in removing the slag in this manner with sufficient rapidity as to necessitate the addition of a second fore-hearth and slag-run to the furnace, on the opposite side from the original opening.

Little need be said regarding the slag-buggies as furnished by the manufacturers of metallurgical appliances, except to urge the necessity of extreme lightness combined with strength. The attachment between pot and axle is rarely sufficiently strong, and the axle itself should be chilled or case-hardened to prevent its rapid destruction where the wheel takes its bearing. The accompanying cut shows the form of slag-buggy adopted by the Orford Company, and is the best and strongest pattern known to the author.

The ordinary complement of slag-pots to a 50-ton furnace is 10, which number should be doubled if it is expected to allow the slag to cool before dumping, as should always be done if possible, in order that the bottom of every potful may be examined for shots of matte.

Great care should be taken to maintain the dump perfectly smooth, which may be easily effected by pouring pots of liquid slag over its entire surface. The sole duty of one experienced workman should be to examine the slag and keep the dump in order, by which means a control of the smelting is maintained and the labor of the pot-runners greatly lessened.



As slag-dumps are not infrequently situated on the margin of a river or lake, the danger of dumping entire pots of slag into water, when they are only partially cooled, should always be impressed upon the workmen. Terrific explosions sometimes result from the penetration of water to the liquid center of a cake of slag that appears quite solid on the outside.

The writer has seen, from this cause, several men badly injured; the iron roof and siding partially stripped from a building 200 feet distant, and an entire town a mile distant alarmed by the explosion of a cake of slag.

BLOWERS AND ACCESSORY BLAST APPARATUS.

All apparatus employed for the production of a blast may be divided into two classes:

I. Those producing a positive blast, and which, if obstructed, must result in the bursting of some part of the apparatus or the stopping of the blower.

II. Centrifugal fan-blowers, which, even if obstructed, continue revolving, consuming much less power than when engaged in actual work, as the air is simply beaten by the vanes, and revolved in the machine itself, without passing out of the pipe.

This distinction is not always clearly appreciated, and serious mistakes in the construction of the plant sometimes arise from a misunderstanding of the properties of the machine which is to furnish the blast.

Such errors can always be avoided by application to a reputable manufacturer of blowing apparatus, as the subject is one to which much attention has been paid by these parties, who are for the most part quite capable of planning and erecting a suitable blowing plant upon a full understanding of the requirements of the case.

Owing to the light blast used by all copper smelters in the United States, the high-priced cylinder blowers, as required in iron-smelting plants, are seldom, if ever, adopted.

The blowers in almost universal use are frequently styled "fan-blowers" in general; but this is a misnomer; for, although it is true that both types of blower in common use resemble a fan in appearance, the results obtained are widely different; one class belonging to the first section, or "positive pressure blowers," while the other is truly a "centrifugal fan-blower," and belongs under the second heading.

To the positive pressure blowers belong the "Root," "Baker," and "McKenzie" blowers, all of which are too well and favorably known to require description or recommendation.

The volume of air delivered by one of these machines can be calculated with great accuracy, while the pressure simply depends on the rapidity of revolution.

From the nature of the apparatus, the wind is delivered in a succession of rapid but distinct impulses and puffs, to which peculiarity the adherents of the apparatus attach a great, although somewhat mysterious, virtue, while its rivals bring forward equally convincing proofs of its damaging effects upon the smelting process.

According to the author's experience with the principal makes of domestic blast machinery, these puffs have neither a damaging nor beneficial effect; an unbroken stream of wind of equal volume and pressure producing exactly similar results.

From the nature of the apparatus, more power must be required to drive its closely fitting parts than if they revolved freely in space, while the cog gearing by which motion is imparted to the piston in most instances produces its disagreeable sound and adds to the number of parts subject to wear.

But the workmanship is so excellent, and the various parts in the modern machines of this type are so admirably fitted and adjusted, that the evils just mentioned are reduced to a minimum; while the nature of the blower is such that only a comparatively slow motion is required to produce the desired effect.

The machines of the second class, or centrifugal fan-blowers proper, consist merely of a light blast-wheel revolving in an inclosure of much greater diameter, and so shaped that wind enters the escape-pipe largely from the centrifugal force acquired by the enormous velocity imparted to it by the fan.

The smaller blowers make from 4,000 to 5,000 revolutions a minute, and those of even 5 or 6 feet in diameter are speeded up to 1,500 or 2,000.

While this is an excellent device for the delivery of a large volume of wind, the element of pressure is only obtained by a great waste of power, the speed necessary to produce a given pressure increasing out of all proportion to the gain in that quality.

The admirable workmanship of the "Sturtevant" and of other kindred fans, and the skill with which the inherent defects of this method of gaining pressure have been reduced to a minimum, have caused the adoption of this form of blower in many establishments where a considerable pressure is required.

But while the writer fully recognizes its usefulness over a very wide range, its fatal defects as a blower for smelting purposes cannot be concealed.

A cupola, for instance, filled to the throat with fine ore and using for fuel a very dense variety of coke, shows evidences of chilling, while the tuyeres become capped with a tenacious slag, which requires a forced blast to keep them open. If the blast is derived from a positive blower, it rises to the occasion, and the pressure of the wind keeps pace with the growing obstruction, as may be seen by observing the gauge. More work is thrown on the engine, and the pressure rises until the obstruction is cleared away, or some portion of the machinery or blast-pipe is overtaxed. (Positive blowers should always be provided with a safety-valve.)

The result is quite different with the fan; for as the wind-stream is obstructed, so is the delivery lessened, until finally, with the complete closure of all blast openings, the work of the engine drops to almost nothing, and the fan revolves at its full speed, neither receiving nor delivering a cubic inch of blast. This may be easily verified by suddenly shutting the main blast-gate between the fan and the furnace. The unaltered stand of the manometer and the sudden decrease of the labor performed by the motive power will sufficiently convince the most skeptical.

With coarse ore, an easily fusible charge, and large tuyere openings, the lightness, compactness, and low first cost may speak in favor of the fan-blower for cupola work.

The Orford Company makes use of the fan even for its enormous furnaces, but expends from 50 to 75 horse-power in obtaining a blast that, according to both theory and experience, could be produced with about one-third of this expenditure of force with the positive blower.*

Where the positive blower is used, it is an excellent arrangement to have engine and blower combined, thus rendering the blast entirely independent of the remaining plant.

Owing to the vastly varied practice in regard to the size of tuyeres used, and consequently in the volume of wind required, it is difficult to give accurate figures regarding the power required for a furnace of any given size.

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* Experiments by Mr. H. M. Howe contradict this statement, as regards power required, but the author is not willing to accept their results in defiance of long years of practical observation.

Elaborate tables, showing the horse-power required under nearly all possible conditions, are issued by the blower manufacturers, and are in many cases quite correct, as controlled by indicator cards taken in the presence of the writer.

As a guide for possible estimates, it may be assumed that eight horse-power will drive a positive blower suitable for a 36-inch furnace, while a 48-inch cupola will require from 12 to 14 horse-power.

The speed required by the fan-blowers is so great as to demand some attention to the arrangement of pulleys and shafting to obtain the same.

Care should be taken to use the largest-sized pulleys practicable, both for driving and receiving, and all abrupt belting from very large to very small pulleys will always give trouble.

A much neglected portion of the blowing-plant is the pipe that conveys the blast. By a strict observance of the following rules, much annoyance may be avoided.

Use galvanized iron, No. 22 to 24. For any length of pipe up to 50 feet, make the blast-pipe 20 per cent. larger in diameter than the outlet of the blower; for from 100 feet to 200 feet, 30 per cent. larger, and if the distance be over 200 feet, make the entire pipe 50 per cent. larger. This precaution will diminish friction and greatly increase the effective blast.

If branches are used, remember that, on account of friction, two pipes of a given area can convey only about three-quarters the wind of a single pipe of their combined area.

Make easy curves, avoiding all angles.

All joints should be riveted and soldered. Remember that the slightest leak may reduce the effectiveness of the blast to an enormous extent.

Have tight-fitting blast-gates in the main pipe and at each tuyere.

Maintain a reliable quicksilver manometer connected with the wind-box surrounding the furnace, and accustom your furnace-men to rely upon it as the seaman relies upon the barometer.

Remember, in this connection, however, that with a positive blower, a high stand of the manometer may indicate that the tuyeres are obstructed as well as that the blower is working satisfactorily.

A heavy boiler-iron damper should always be fitted in the cupola flue or down-take, and should be lowered to just such a point that the fumes escape lazily, but without issuing from the charging-door. By this precaution, a diminution of 90 per cent. in the amount of flue-dust may be effected—by accurate and long-continued trial.

MODERN AND ACCESSORY BLAST-FURNACE APPARATUS.

While the slag from nearly all American copper cupolas is run into movable pots and thus conveyed to the dump, the more valuable product is handled in several different ways, according to its richness and to local custom.

Furnaces containing an inner crucible or an exterior fore-hearth, in which the metal collects in considerable quantities, and where no metallic copper is produced, are usually tapped into sand-molds, made of a slightly moistened sandy loam. With proper management, this method is economical and satisfactory, and suited to almost any grade of metal; but to avoid the untidiness resulting from the presence of a large quantity of sand in the neighborhood of the furnace, as well as the frequent mixing of the same with the liquid matte from careless manipulation, a series of heavy cast-iron molds, communicating with each other by lateral projecting lips, is sometimes preferred. It is necessary to warm them thoroughly just before tapping, to prevent injury from the enormous temperature to which they are exposed.

The Western water-jacket furnaces are almost invariably provided with two iron spouts—one in front, for slag; the other, at the back or side of the furnace, and several inches below the former, for metal.

The slag is tapped into pots at intervals of from five to fifteen minutes, by piercing a clay plug with a light, pointed steel bar, while the metal collects until the space between the two spouts is filled, when it is tapped into rectangular iron molds, with tapering sides and ends, and mounted on wheels. The ordinary weight of the pigs thus produced is from 250 to 400 pounds. The method is convenient and cleanly, though a considerable expense arises from the rapid destruction of the metal molds.

In furnaces provided with a siphon-tap, and in which the molten products flow continuously, the matte is usually received into ordinary slag-pots, and allowed to stand until it is chilled sufficiently to be dumped on the floor of the building. This practice not only requires a large number of pots, but also leaves the matte in a peculiarly massive and unmanageable condition, some of the lower grades of metal produced from basic ores being almost malleable from the excess of iron present. It is at times hardly possible to break them with a heavy sledge, and their subsequent crushing is ruinous to the machinery employed.

On this account, the old plan of pouring the liquid matte on a cold iron plate is sometimes adopted, and with very satisfactory results. Heavy plates of cast-iron are used, 30 by 55 inches, and from 2 to 3 inches thick. These are inclosed by a tapering border, some 3 inches high; and the liquid matte, when poured upon them, spreads out at once into a thin sheet which is easily broken up into sizes suitable for immediate stall or heap-roasting, and in excellent condition for pulverization.

An ingenious improvement in connection with "tapping" has been introduced into the Grant Smelting Works, and other Colorado establishments, and although intended for furnaces smelting the precious metals, may be applied also to copper furnaces where the manner of tapping is such as to call for it.

When copper or lead matte containing the precious metals is tapped into an iron pot or basin, a considerable quantity of slag gushes out at the close of the operation. This, owing to splashes of metal, which have spattered over the sides of the pot to a considerable height, becomes so rich in silver (and gold) as to cause too great a loss if thrown away, while it may yet be so poor as scarcely to repay another fusion. To obviate this difficulty, at some works, the tapping-pot is provided with several plugged openings, at about the point corresponding with the lower surface of the slag which floats upon the metal. After tapping, it is allowed to cool for a few moments, when the lateral plug is pierced, and the supernatant slag flows out in a liquid state, leaving, however, a thin crust covering the walls of the pot, the cold iron having rapidly chilled the slag with which it was in contact, and which also contains all the metal that was splashed against the basin in tapping. In this manner, a few pounds of slag are obtained, assaying from 12 to 15 ounces of silver to the ton, in place of some hundreds of pounds containing only 4 or 5 ounces of the precious metal.

MATTE SMELTING IN BLAST-FURNACES.

To one accustomed to the smelting of copper ores in blast-furnaces, the fusion of roasted matte presents no difficulties, as the homogeneity and fusibility of the material, coupled with the absence of silica, alumina, and most other refractory substances, relieve the operation of its chief difficulties.

A single examination of a proper sample of the calcined material reveals the proportion of oxide of iron—and other bases—present, and determines the amount of siliceous flux that must be added to produce a proper slag.

As the slag resulting from the fusion of matte almost always contains enough copper to demand its further treatment—owing to the extreme richness of the product—it is usually advantageous to add the smallest possible proportion of siliceous flux compatible with the formation of a reasonably pure slag. In this way, a slag is obtained that is so rich in iron as to be of great value as a basic flux in the smelting of siliceous ores. No material generally accessible to the metallurgist is more fusible or more useful in correcting a faulty mixture or in clearing out a choked furnace than the one in question.

If carbonate or oxide ores of a siliceous nature are available, the entire aspect of the smelting process is changed for the better, as they can be used as a flux for the roasted matte, while, being free from sulphur, their copper contents assist in producing a concentrated matte of higher percentage than would otherwise result.

The last-mentioned benefit is so great that it is usually profitable to purchase such ores even at a price so high as to leave no margin after deducting the working costs.

Owing to the great distances between most of the establishments engaged in smelting sulphide ores and the mines producing siliceous oxidized ores in any considerable amounts, but few instances occur of such a happy condition of affairs; and therefore in most cases, gravel, pebbles, and other barren substances are used as a flux to the matte.

When forced to adopt this unfortunate practice, experience has shown that more economical and better results are obtained by the employment of some acid compound—such as clay slate, silicate of alumina with a small proportion of other bases, etc.

—than by using pure silica, in the shape of quartz pebbles, crushed quartz, sand, rock, etc.

The extreme infusibility of the latter requires a much higher temperature for its combination with the protoxide of iron than do the less refractory substances named. A pretty thorough trial of almost every available material has led to a preference for ordinary mica-schist or clay slate; the next best substance is broken common red brick, the form of the latter rendering them superior to the clay from which they are made.

In all cases, the siliceous flux should be broken to the size of walnuts before use; otherwise, irregularities in running may be anticipated, especially in small furnaces. Minerals containing any considerable proportion of silicate of magnesia should be avoided.

It seems hardly necessary to mention that the simple fusion of *unroasted matte* in a cupola furnace produces practically no result except a change of form, the removal of sulphur by sublimation being so slight as to cause an enrichment of only one or two per cent.

Although this subject has been already briefly discussed, there exists such a widespread idea among non-professional men that a mere fusion of the matte is sufficient to increase its value that a positive statement to the contrary, accompanied with some experiments which were executed to demonstrate this fact to a doubting director of a smelting company, may prove of value in some future instance. Twenty-six tons of carefully sampled unroasted matte, broken to the size of an egg, were smelted in ten hours in a cupola furnace with a wind pressure of one inch mercury. Ten per cent. of ordinary ore slag was added, to protect the metal.

On accurately weighing and sampling the product of fusion, the following results were obtained:

Weight of matte smelted	26·00 tons.
Weight of slag smelted	2·60 tons.
Assay of matte smelted	33·50 per cent.
Assay of slag smelted	0·37 per cent.
Weight of matte produced	25·07 tons.
Assay of matte produced	34·70 per cent.
Assay of slag produced	0·32 per cent.

Showing an enrichment of only 1·2 per cent. The amount of copper produced differs only 0·001 per cent. from the amount charged, showing a remarkable agreement in assays, weights, etc.

Further experiments were made with corresponding results, although with more variation in the figures obtained.

Notwithstanding the fusibility of the mixture, fully as much fuel is required in smelting roasted matte with its siliceous flux as in the case of calcined ore, one pound of good coke being required for from 7 to 9 pounds of charge. This may result from the great quantity of reducing gases required to lower the sesquioxide of iron, which is usually largely present in roasted matte, to a protoxide, as well as to the greater tendency of metallic copper or very rich matte to chill in the bottom of the furnace, which must be counteracted by an additional quantity of fuel.

The same pattern of furnace used for smelting ore is also applied to the concentrating fusion of matte, the height of American copper cupolas seldom being so great as to unfit them for this work.

The richness of the product depends on the thoroughness of the calcination, as well as on the extent of the reducing action in the smelting process.

It is seldom that the calcination has been executed so thoroughly as to yield solely metallic copper in fusion. A variable quantity of rich matte accompanies the metal. The grade of the copper may be also unduly reduced by metallic iron, as was the case in the Houghton slag smelting, where the anthracite used as fuel caused a large and unwelcome adulteration of the copper product with iron.

This is, however, unusual in American practice, the lowness of the furnaces and the rapidity of the process combining to produce a tolerably pure metal, as will be seen from the following partial analyses of pig-copper resulting from the fusion of roasted matte in the cupolas:

Ely pig-copper (by Nolten):

Iron	1·6
Sulphur	0·8
Copper	97·2
	99·6

Ely pig-copper (by Peters. Selling sample of 200,000 pounds):

Iron	1·1
Sulphur	0·7
Copper	98·4
	100·2

Ore Knob pig-copper (by Griffith):

Iron	1·4
Sulphur	1·1
Copper	96·8
	99·3

A comparison of the foregoing determinations, which represent the average condition of very large quantities of black copper as produced in this country from the fusion of roasted matte, with the following analysis from Percy of the average black copper produced at Atvidaberg, Sweden, will show the advantages resulting from a more rapid execution of the process and other improvements:

Copper	94·39
Iron	2·04
Zinc	1·55
Cobalt and nickel	0·63
Tin	0·07
Lead and silver	0·30
Sulphur	0·80
	99·78

An average sample of pig-copper from the Detroit and Lake Superior Smelting Company's cupolas at Houghton contained only 94 per cent. copper, the remainder being principally sulphur and iron. This extreme impurity, from such remarkably clean metallic ores, arises from the sulphur in the anthracite used as fuel, and the excessively powerful reducing action, especially when it is charged in the ratio of about one pound to each four pounds of material smelted.

The presence of zinc, cobalt, nickel, etc., in Dr. Percy's sample of Swedish copper will account in part for the low grade of the black copper, but the principal reasons for the difference in quality are those already mentioned.

Three samples of black copper from the celebrated Mansfeld works in Prussia, made respectively by Berthier, Hoffman, and Ebbinghaus, contained 95·45, 89·13, and 92·83 per cent. of copper; while two determinations of the same material from the Riechelsdorf smelting-works, in Germany, by Genth, give respectively 83·29 and 92·24 per cent. of metal.

The influence exercised on the succeeding operations by the purity of this product is very great, the lower grades of black copper requiring one or more oxidizing fusions to bring them to the same purity as that already possessed by the immediate product of most American furnaces used for the production of pig-copper.

The product of matte concentration in blast-furnaces differs from that derived from the same process when executed in reverberatories in not being homogeneous, but consisting usually of a matte of medium high grade, together with a certain proportion of metallic copper, where, in the latter case, it would consist entirely of a matte of very high grade. This is a most interesting fact, and yet awaits a satisfactory explanation.

The large brick furnaces already described are also used with advantage for matte concentration, their principal drawback being the inevitable tying up of a large quantity of metal in the bottom of the cupola. This deposit increases according to the quantity smelted, and even in a well-constructed furnace may amount to 20 tons or more. This drawback will doubtless be eventually overcome, but for the present prohibits the employment of such cupolas for the purpose indicated for any but very large metallurgical concerns, which can afford to submit to the locking up of such a large amount of metal for the sake of the economical advantages belonging to this type of furnace.

TREATMENT OF FINE ORE IN BLAST-FURNACES.

The mechanical condition of the ore to be smelted in blast-furnaces is a matter of scarcely less importance than its chemical constitution.

The evils resulting from an undue proportion of fines are well known.

The formation of an immense quantity of flue-dust is one of the least of these evils, as provision can be made for its collection and reworking, though at an increased cost; but the difficulties resulting from the choking of the furnace, and the sifting of the fine ore through the charge until it pours out in a stream through the tuyere-openings, scarcely altered by its passage through the furnace, are radical, and incompatible with either proper or economical work.

The extent of this evil has encouraged the invention of a great variety of methods for its removal, most of them relating to a consolidation of the fine material into lumps of a suitable size.

The agglomeration of the fine ore in the calcining-furnace has been suggested; but the great expense of fuel and the heavy losses inseparable from a method that, however applicable to such an easily fused substance as silicate of lead, would be entirely impracticable when dealing with oxide of iron, render it unnecessary to discuss this practice.

Assuming that the only feasible remedy consists in forming the fine ore into blocks, the experiments executed naturally fall into three divisions:

1. Bricking by the aid of some foreign substance that has the power of holding the ore particles together.
2. Bricking by pressure alone.
3. A combination of the two methods.

The materials tried by the writer and included under the first heading are: Silicate of soda (soluble glass), unslacked lime, clay, hydraulic cement, coal-tar and similar substances, sulphate of iron.

In nearly all cases, a certain degree of pressure must be used to form or mold the mixture into the desired shape; this may be obtained by an ordinary brick-machine, or by compressing with the hands, using a mold or not. In No. 2, pressing alone is used. A thorough mixture of the ore with silicate of soda results merely in the coating of each particle with a layer of soluble glass, and in nowise facilitates the agglutination of the ore. On the other hand, when the latter is already compressed into balls or blocks, the dipping of the same into a strong silicate of soda solution is accompanied with great advantage, the surface becoming, on drying, nearly as hard as granite, and effectually preventing any wastage or breakage of the lumps by handling. (It should be mentioned that the circular or oval shape is much preferable to the rectangular, owing to the absence of fragile edges and corners.)

Of course, this material would be far too expensive for anything but the richest ore, sufficient water-glass to thoroughly coat a ton of balls the size of the fist costing, at Eastern wholesale prices, about $3.25.

No substance has been more frequently employed for the purpose indicated than freshly burned lime, which should be slacked with considerable water, and the resulting milk of lime thoroughly incorporated with the ore, until the entire mass possesses the consistency of very thick mortar.

This is usually left in a heap for several days, and then fed into the furnace in the shape of partially dried mud. But much better results are obtained by forming it at once into balls and subjecting it either to the hot sun or to a gentle artificial heat until it is thoroughly dry and hard. The resulting balls are somewhat brittle and fragile, and demand careful manipulation; but are far preferable to the product obtained by leaving it in a heap, and exert a marked effect in the capacity and condition of the smelting-furnace.

The proportion of lime necessary to effect a good result varies greatly, according to the physical condition of the ore, the amount of sulphates present (which form a strongly cohesive cement with the lime), etc., but is usually from 5 to 12 per cent.—less than 5 per cent. seldom producing satisfactory bricks. The cost of mixing alone (lime not included) is from 25 to 40 cents a ton by contract, which sum must be doubled or trebled if it is formed into bricks, depending upon the effectiveness and convenience of the plant. In almost all cases, the addition of lime has a favorable effect upon the subsequent fusion. It is probable that, when the water is removed from the lime by the heat of the furnace, the masses again crumble to a certain extent, but not until they have already undergone a certain preparation, which must be of value, to judge from the results obtained in actual work. This method was carried out extensively at the "Gap" nickel mine, Pennsylvania, not only the roasted fines, but also the fine raw pyrites being thus treated, previously to roasting in kilns; the results of the latter process being much better than could be expected from a material possessing such slight cohesive properties as fine granular pyrites.

The Orford Company and many other metallurgical establishments have adopted this method in the handling of finely pulverized, calcined matte, although in most cases the materials are simply mixed into a thick mortar, and charged into the furnace after lying in a heap for a few days. The difficulties and irregularities in the running of the cupola that would certainly result from the employment of an excessive proportion of such unfit material are counteracted to a considerable extent by the addition of a large amount of slag, which serves to loosen the charge and keep everything in normal condition.

The important observation has also been made by Mr. W. E.

C. Eustis, that a product of considerably higher grade results from the addition of some 5 per cent. of lime to the calcined matte. The substitution of limestone fails to produce the same effect.

In consideration of the advantages already enumerated, and from the fact of its cheapness, general availability, and fluxing qualities, lime may be regarded as the most useful substance yet known for the purpose under consideration, and where bricking under pressure, with subsequent thorough drying, is not attempted.

Clay is also extensively used for the same purpose, and if thoroughly incorporated with the fine ore and allowed to dry for a reasonable time after being made into balls, gives a stronger and less friable product than lime. The quantity added varies from 2 to 5 per cent.

It possesses the serious disadvantage of adding to the siliceous contents of the ore. In the case of calcined matte or highly basic ores, on the contrary, it forms a useful flux. The cheapest variety of clay that possesses the required plasticity should, of course, be selected.

The powerful cohesive qualities of ordinary hydraulic cement long since attracted notice.

Fortified by the favorable opinion of Prof. J. Fraser Torrance, the writer has employed it to brick jewelers' sweeps, and after a month's trial, is quite satisfied with the results obtained.

He finds about eight per cent. of cement necessary to produce balls which, after a week's exposure to the air, will bear moderate handling, and give good results in the furnace. Of course, the expense of this method forbids its use for ordinary substances.

Where coking coal is available, fine ore can be mixed in large proportions with the coal in the kiln and coked.

Coal-tar and similar substances require the aid of quite powerful compression to answer the required purposes, and have not been found practicable.

A solution of copperas—sulphate of iron—is used in several of the European works to agglomerate fine ore. By careful drying, the balls made with this substance become very hard; but the addition of sulphur to the charge (forming a perceptible increase of matte in cupola work), the very disagreeable effect upon the skin of the operatives, and other minor disadvantages, have prevented its adoption.

The introduction of inexpensive machines for the manufacture of brick from almost dry clay, and capable of exerting an immense pressure, has opened new possibilities to the metallurgist. Although, doubtless, such exist, the author can find no recorded results of bricking fine ore by employing pressure alone, and is therefore obliged to fall back upon some brief trials made under his directions at the Parrot Works, Butte, Montana. The ore used consisted of pyrites concentrates, calcined so thoroughly as to contain only traces of soluble sulphates. The brick-machine used produced about 40 bricks a minute, weighing 5 pounds each dry, and exerted a pressure of 4 tons per square inch. Under this immense force, the compressed ore slabs already possessed considerable strength, and could be backed up in the usual manner.

Unfortunately, no provision had been made for drying the brick by artificial heat—a most essential part of the process. After a day's exposure to the air, they were smelted in a water-jacket furnace, breaking up to a considerable extent during transportation, but fusing with much greater rapidity and economy than when in a fine condition.

A few that were dried at a gentle heat for six hours became so hard as to bear any reasonable handling, and when broken once in two, were admirably adapted for blast-furnace work. Rapid drying is highly injurious.

The writer is quite convinced of the value of this method, and considers it applicable to any ordinary material.

The essential conditions, after obtaining the proper pressure, are a gentle and sufficiently prolonged temperature, and a sufficient space to dry the necessary quantity. A series of light shelves in a well-ventilated building, heated by steam-pipes, would seem to fulfill these requirements, while the shape and size of the molds could be adapted to the purpose. A round or oval shape is best, thus escaping the wear on sharp corners and angles.

The cost of bricking fine ore in this manner in Montana did not exceed 50 cents a ton; a single machine, requiring 10 horse-power and the labor of 8 men, having a capacity of 60 tons in ten hours.

A combination of the two foregoing methods was effected by incorporating a certain proportion of lime or clay with the fine ore, before submitting it to the immense pressure mentioned.

The addition of from 2 to 4 per cent. of either of these substances was accompanied with an increase in the strength and tenacity of the product, and was found especially useful where the process of drying could not be carried out. With proper facilities for a slow but perfect desiccation, no such addition is necessary.

Mr. O. K. Krause, President of The Vermont Copper Co., has obtained excellent results by stirring green fines into the molten slag from his black copper furnaces. From 75 to 100 per cent. of the weight of the slag can be thus stirred in, the cooled mass being broken up and resmelted.

In place of clay, fine slimes from the concentration department or other sources may be substituted, and their metal contents beneficiated at the same time. This practice was adopted by Prof. J. A. Church, at Tombstone, Ariz. An ordinary brick-machine was employed, the cohesive property of the slimes being depended on to bind the fine ore together.

Fine grinding has been lately proposed: it forms a pulp, which becomes tenacious from the minuteness of its particles. This plan is widely practiced in England for the balling of the raw Spanish pyrites fines, preparatory to their roasting in kilns. The tenacity generated in this otherwise granular and uncohesive material by a mere grinding is very striking.

Before concluding this subject, the question of smelting fines in their natural condition should be noticed.

While the presence in a cupola smelting charge of even a moderate percentage of fine ore is accompanied with certain evils, such as formation of flue-dust, the choking of the furnace, irregularities in its running, descent of the fine ore unprepared, until it even pours out of the tuyeres, scarcely heated above the temperature of the air, etc., it is a condition that is almost invariably met with to a certain extent, as the mere transportation of ore from one building to another will result in the formation of a certain amount of fines. It becomes important, therefore, to determine at what point the proportion of fines becomes so great as to demand measures for its relief.

This again varies greatly with the quality of the ore, slag, and metal, the power of the blast, size of furnace, capacity and efficiency of dust-chambers, etc.

Here, as in most other instances, no experiments have been recorded to determine this important point, and practice varies with the prejudice or opinion of every individual.

The following experiments were made on ore from the Moose mine, in Park County, Colorado, in 1871, and though relating to the treatment of silver ore, will serve the present purpose as well as though the product had been a copper matte. The furnace was small—2½ by 3 feet—and smelting an exceedingly infusible charge, containing over 40 per cent. of sulphate of baryta and much silica.

The fuel was spruce charcoal, and the blast very weak, causing an extremely slow smelting; but as the conditions remained the same throughout all the experiments, the results possess some value.

The material smelted consisted of ores from the Moose and adjoining mines, from which all that would pass through a 3-mesh screen had been separated, to be submitted to a calcination, the fine ore containing a much greater proportion of sulphides than the coarse, which latter was smelted raw.

To this coarse ore was added a certain amount of carbonate of lead ores, almost free from fines and the requisite quantity of heap-roasted auriferous iron pyrites, to produce a fusible slag. The latter material contained, after roasting, about 25 per cent. of fines; and the mixture as prepared for the furnace, and without the addition of the fine calcined silver ore, carried about 12 per cent. of fines that would pass a 6-mesh screen.

This was regarded as the normal charge, to which, by way of experiment, were added varying proportions of the roasted fines; all other conditions remaining as nearly identical as was practicable throughout the trials.

Representing the quantity of this normal charge that would be smelted in twenty-four hours by 100, the addition of fines produced the following decrease:

10	per cent.	fines reduced	it to	92
20	“	“	“	80½
25	“	“	“	80
30	“	“	“	64
35	“	“	“	56
40	“	“	“	51
50	“	“	“	42

Aside from the decrease in capacity accompanying the addition of fines, serious irregularities in the running of the furnace were also produced, causing an increase in the cost of smelting per ton, as well as greatly adding to the labor of the men and to the proportion of silver lost in the slag.

An increase in the area of the furnace greatly heightens its capacity for smelting fine ore; and the results obtained in this direction by the use of the large Orford furnace are very striking.

The presence of from 15 to 20 per cent. fines seems to be no drawback at all, when this type of furnace is employed, and even from 50 to 60 per cent. of the charge may consist of this ordinarily unwelcome substance without seriously affecting the running of the furnace, although, of course, its capacity will be somewhat reduced.

After smelting a charge containing a very high proportion of fines for from 24 to 36 hours, it will usually be found that cold, unaltered fines appear at the tuyeres. This results from the constant agitation of the charge by the blast, by which the ore particles are sifted down through the interstices between the fuel and coarse ore, until they actually reach the level of the hearth. In such cases, the pipes should be removed and all the fine ore within reach raked out of the tuyere-holes, with appropriate tools.

All feeding from above should now cease until the charge, under a light blast, has sunk nearly to the tuyere level, when the shaft should be refilled with alternate layers of fuel and *coarse* ore in the usual proportions, after which the use of fine ore may be resumed. With these precautions, a very large proportion of fine ore may be smelted in this type of furnace, without seriously diminishing its capacity or producing any irregularities of importance.

The attention of metallurgists is particularly called to the ease with which raw fine pyrites, technically called "green fines," may be treated in this furnace with the fan-blast. This material often accumulates in great quantities at copper mines, as, owing to its mechanical condition, it cannot be roasted in heaps, while pecuniary considerations may forbid the erection of an extensive calcining plant for its treatment. Heretofore, when of low grade (from 1½ to 3½ per cent.), it has been either thrown aside in heaps, or allowed to harden and consolidate until it can be broken out in lumps and added to the roast-heaps. By this practice much waste occurs, while a large amount of money is constantly tied up in this material.

After experimenting, Mr. J. L. Thomson, of the Orford Company, found that a charge composed of this material and ferruginous slag from the concentration-fusion of copper matte could be smelted together to great advantage in the large furnace; the immense volume of blast employed oxidizing the raw pyrites to a considerable extent, and producing a matte of much higher grade than would result from such material under ordinary circumstances. The slag from matte concentration always carries nearly or quite enough copper (from 1 to 2 per cent.) to cover the cost of its resmelting where fuel is cheap; while the large percentage of protoxide of iron that it carries neutralizes the silica of the green fines, which finds no base in its own composition, all the iron that it contains being combined with sulphur, and consequently unavailable for slag formation.

Where circumstances do not favor the employment of the matte slag, this may be replaced by heap-roasted pyritic ore; in which case, of course, the resulting matte will be somewhat richer.

In any case, the slag resulting from this practice is distinguished by its freedom from copper, owing to the overwhelming amount of low-grade matte present, which cleanses the light siliceous slag to an unprecedented degree. Owing to the large amount of unsatisfied silica in the green ore, the slag is always exceedingly acid, containing from 48 to 55 per cent. SiO_2, and often being so sticky and thick that only the constant and powerful stream of intensely hot, low-grade matte keeps the slag-run open, and prevents the furnace from "sticking up."

The oxidation of the fine sulphide particles by the air-blast carries the heat to the surface of the charge, and produces to a certain extent those evils inseparable from the extension of the high temperature from the zone of fusion to the upper layers of the charge. Owing to these circumstances, a serious burning of the brick walls often takes place, which circumstance, combined with the cutting down of the furnace bottom from the immense quantity of fiery, low-grade metal, favors the practice already recommended of keeping in blast only during twelve hours out of the twenty-four.

The siphon-tap is almost indispensable in this form of smelting, as the quantity of matte produced in a twenty-four hours' run often amounts to 25 or 30 tons.

From six to nine months has been the ordinary length of campaign for these large brick furnaces, running on either a siliceous or basic slag, at the expiration of which time a week's repairs will again fit them for work.

Not feeling at liberty to give the results obtained in the fusion of green fines at the Orford Works, where this practice originated, the author is forced to fall back upon results in his own practice, where the total quantities treated, though much smaller, still aggregate some 30,000 tons. During a four months' campaign, in which a mixture of green fines and matte slag were treated in a large Orford furnace, the average daily (twenty-four hours) results were as follows:

Weight of green fines smelted	49·71 tons.
“ “ matte slag “	38·40 “
Total	88·11
Gas-coke used	16·10 tons.
Assay of ore	3·87 per cent.
Assay of matte slag	0·94 “
Weight of matte produced	20·67 tons.
Assay of matte produced	10·70 per cent.
Assay of slag produced	0·13 “

Rate of concentration, about 2½ tons of ore into one. The copper produced in this campaign, after taking into consideration the metal gained in the matte slag, and deducting the small amount lost in the slag from the operation itself, agreed almost exactly with the amount calculated from the careful and frequent assays made.

The matte produced, though very low in grade, is roasted in heaps with great facility, and forms a most welcome flux for siliceous ores.

This practice is unique and well worthy of attention.

The size to which ore must be broken for cupola smelting depends upon two factors—its fusibility and its conductivity.

Fusible material, especially if porous—such as ferruginous and calcareous ores, basic slag, etc.—may be charged in fragments from 3 to 5 inches in diameter without producing evil results, although good practice demands its pretty uniform reduction to the size of a large apple (that is, fully 3″). The same may be said of fragments of copper matte or metallic substances, which, being excellent conductors of heat, melt all at once, where a piece of quartz or fire-clay might be in a state of fusion on the surface, while hardly heated at the center, and consequently should be invariably reduced to the size of horse-chestnuts, unless smelted in company with a large proportion of basic ore. A striking example of this may be found in the fusion of "mass copper" at Lake Superior, where pieces of this metal weighing five or six tons are smelted on the hearth of a reverberatory furnace with no difficulty or delay, the high conductive power of copper causing an equal distribution of heat and simultaneous fusion of the entire mass. A rock of the same size could never be smelted except by the gradual wearing away of its exterior surface.

The following résumé is from a paper on Copper Smelting by H. M. Howe, in the Bulletin of the United States Geological Survey.

RÉSUMÉ.

To sum up, for SMELTING ORE the *cupola* is especially advantageous—

I. With highly ferruginous ores.

II. Where the cost of anthracite, coke, or charcoal is not excessively greater than that of bituminous coal, wood, and other fuels fitted for the reverberatory only.

III. For oxidized ores.

IV. For low-grade native copper.

V. Where, as in the case of lean ores, clean slags are a necessity.

The *reverberatory* is especially advantageous—

VI. With highly refractory siliceous, aluminous, calcareous, or magnesian ores.

VII. Where the composition of the ore changes suddenly and greatly.

VIII. Where bituminous coal, wood, or other reverberatory fuel is very much cheaper than anthracite, coke, or charcoal.

IX. For smelting and immediately refining rich native copper.

X. Its disadvantage in yielding richer slags than the cupola weighs less heavily in case of rich ores.

CHAPTER XII.

LATE IMPROVEMENTS IN BLAST-FURNACES.

THE principal changes have been in the general arrangement of the smelting plant, in improving certain details, and in effecting some slight economies.

Herreshoff has designed in connection with his furnace an excellent charging platform, consisting of flanged cast-iron plates bolted together, surrounded by a rail of ordinary one-inch gas pipe and supported by girders in such a manner that there are no posts to interfere with the work about the furnace.

Finding that the cast-iron plates forming the floor of the furnace-shed soon become warped and destroyed by the stream of hot slag falling upon them every time that the slag-pots are changed, I have remedied this difficulty by cutting a circular hole sixteen inches in diameter in the plate, directly where the slag stream falls, flanging the same, and providing it with a cast-iron basin, which is kept filled with sand and can be changed at will; the wheels of the slag-pot straddle this basin so that it does not interfere at all with the work.

When running on a low-grade copper matte, and more especially if producing a matte containing nickel, the bronze tapping ring used in connection with the Herreshoff fore-hearth soon becomes porous and leaky, so that if not changed at once, the liquid matte is certain to eventually get into the water space and produce a dangerous explosion. No amount of care in casting this bronze alloy seems to help matters. The addition of a small percentage of silicon was tried, but neither did this, nor the addition of aluminum, help matters.

Molten tin and solder were poured into the bronze casting, and it was then shaken until the interior was plated with a thick layer of this malleable metal, but without avail. The minute holes that appeared in the tapping ring could neither be plugged nor corked, as the metal became brittle and rotten, and crumbled under the slightest blow.

At one time, when running some one hundred and thirty tons of ore a day, and producing twenty tons or more of matte containing some 18 per cent. nickel and 20 per cent. or more of copper, it became necessary to change the tapping rings every three or four days, owing to the trouble just mentioned.

After much experimenting it was found that, by reducing the quantity of feed water to a minimum, and thus keeping the tapping ring constantly at a temperature of nearly 200° Fahrenheit, the trouble was remedied to a great extent; and this method has been pursued ever since.

Of course, whenever a pot full of matte is tapped off, it is necessary to let on a little more feed water for the moment.

Another very important precaution to take, especially when running furnaces with a narrow water space, where the feed water is not absolutely pure and free from every trace of sediment, is to provide the furnace with additional discharge pipes at all points where the heat is great and the circulation of the feed water is liable to be sluggish or impeded by sediment or lime salts; such places are the very narrow space between the main outlet hole of the furnace and the iron ring that forms the lower border of the jacket, and also that between the main tap-hole casting in the furnace proper and the same lower border.

A half-inch pipe screwed into a hole tapped through the lower ring and discharging into the same funnel as the waste pipes from the fore-hearth and tap-hole casting, and provided with a valve to regulate the flow of water, may prolong the life of a water jacket for many months.

The cast-iron fore-hearth or *well* used in connection with the Herreshoff furnace is, according to my experience, very liable to crack long before it is worn out. This is especially the case in cold weather, when a furnace is frequently cooled down and started up again. Such cracks can usually be temporarily repaired by covering them with red lead, or with a mixture of salammoniac and iron filings over which a plate of wrought-iron is to be accurately fitted and tightly screwed down. But I have found that in the end a *well* made entirely of wrought-iron is more economical and satisfactory. I have had a separate water-jacketed slag-spout made for these wrought-iron *wells*, as it makes their construction so much simpler.

As the matte when tapped often streams into the pot with such force as to cause waste, it has been found advantageous to diminish the pressure in the fore-hearth by letting the blast escape into the air during the actual operation of tapping. This is most easily accomplished by cutting a large opening in the blast pipe close to the furnace and providing it with a hinged door that is kept tightly closed by a weight, except during the moment of tapping, when it is opened by pulling a wire.

When 100 to 150 tons of slag are run out of a copper furnace in a single day, it becomes not only difficult to handle it in pots, but the wear and tear of the pots amount to a very heavy item. The small scale on which copper works are usually built, prevents the adoption of the methods used at handling slag at the great iron furnaces, where it is usually flushed off in great quantities and at considerable intervals of time, instead of running in a continuous small stream. Where water is obtainable, no better method can be adopted than to granulate and remove the slag by means of a stream of water. A considerable pressure is required, and the secret of using this method with success is to strike the falling stream of slag with such a forcible stream of water as to granulate it instantaneously and sweep it off over the dump. A stream of water three inches in diameter under a head of ten feet should easily handle the slag from a 150-ton furnace, and would require a fall of nearly an inch to the foot.

Where the roasting has been imperfect and it is more a question of obtaining a tolerably rich matte than of putting a large quantity of ore through the furnace, it is possible to drive off much of the remaining sulphur by running with only two or three feet of charge above the tuyeres and using a very light blast. This naturally means a high consumption of fuel, but will produce a very rich matte. It is by this method that the German black copper furnaces and the small copper furnaces at Ely, Vt., produce a large proportion of very pure black copper from an imperfectly roasted, 30% matte. That it is the slow smelting that accomplishes this removal of the sulphur is well shown by the following amusing, but very instructive incident:

At a certain smelting works where the copper furnaces just at that time were running on a very low-grade matte, that was also very badly roasted, the manager suddenly received a visit from several directors of the company. Naturally wishing to make a good showing at the furnaces, he sent out the order to throw half a dozen pigs of black copper into each of the cupolas and put on a heavy blast. In the course of half an hour, deeming that the dose would have its effect, he arrived opposite the copper furnaces with his visitors, and ordering them to be tapped, confidently awaited the metallic flood that was to astonish the directors and lubricate the finances of the company. But instead of copper, the only result obtained was an immense tap of matte of medium grade. The manager's chemistry was so deficient that he had failed to realize the fact that little or no sulphur would be volatilized during such rapid smelting, so that it was bound to combine with such metallic copper as it could find in the furnace, thus not only producing no copper from the original charge, but actually "throwing back" into matte all his extra addition of metallic copper; so that whereas slow running on a poor charge will produce a large proportion of pig copper, rapid running on a rich charge may produce nothing but a matte of tolerable grade.

CHAPTER XIII.

THE SMELTING OF PYRITOUS ORES CONTAINING COPPER AND NICKEL.

THE occurrence of a nickeliferous pyrrhotite containing a more or less plentiful admixture of chalcopyrite seems peculiar to the Huronian and Laurentian systems, and has been so extensively discovered in Northern Ontario, and so largely worked both for the copper and nickel it contains, that, in view of the interest that is at present shown in these mines, and the intimate connection that their treatment has with the general metallurgy of copper, it seems proper to introduce a short description of the most approved methods now employed for treating these ores.

As it fell to the lot of the author to assist in opening the first mines and build the first smelting works in the district referred to, and manage them for a couple of years, he is enabled to describe these matters without deviating from the policy pursued throughout this work of speaking authoritatively of only such matters as have come under his personal observation and within the sphere of his own practice.

The geology and mode of occurrence of these ores have no bearing upon their treatment. Suffice it to say that the nickeliferous pyrrhotite occurs mainly in lens-shaped bodies situated in diorite dikes that cut through the country rock for a long distance, but that only carry bodies sufficiently concentrated to be of commercial value at certain points where the local conditions have been favorable for its deposition.

The pyrrhotite occurs in a very massive form usually, and seems to have a portion of its iron replaced by nickel without the proportion in which the latter metal occurs affecting the physical structure or appearance of the pyrrhotite. The amount of nickel in the pure pyrrhotite varies from 1½ to 9 per cent., averaging perhaps 4 per cent. But the admixture of diorite, chalcopyrite, and other foreign matter reduces the ores treated on a large scale to perhaps 2¾ per cent. of nickel. There are exceptional ores that carry 15, even 20 per cent. of nickel, and some few crystals of pure Millerite (sulphide of nickel) have been found in depth, but these rare exceptions have no effect on the general average.

The copper content of the ore is a foreign admixture, and occurs solely in the shape of a very pure chalcopyrite, carrying very nearly its proper theoretical proportion of copper (about 34 per cent. when pure). This is at times scattered throughout the pyrrhotite in specks and thread-like veins; at other times it occurs in nodules of considerable size, while rarely it is found in almost massive condition, between the vein of pyrrhotite and the wall-rock.

On an average, the ore treated at the principal mines carries some 4 per cent. of copper, though if selected, it could easily be brought up to 8 or 10 per cent. But up to the present time, experience has shown that it pays about as well to mix all the ores and smelt them as they are, as to try to make two grades of matte, one rich in copper and poor in nickel, and the other high in nickel and low in copper. By pursuing the latter course, a slightly better price can be obtained from the refiners, but the metallurgical operations are seriously embarrassed, as, if the heavy nickel ore is smelted alone, it produces far too basic a slag; while if the richer copper ore is fused by itself, the slag is too siliceous to smelt easily. By mixing the two varieties of ore in their proper proportions, a good slag is obtained without the addition of flux, not a pound of the latter being used during the time I was in charge of the work. Besides, the ores roast much better when mixed than if separate.

ROASTING.

The ore, after being run through jaw-breakers at the mines, set to about $1\frac{3}{4}$ inches, is run in railroad cars to a high trestle, which extends the entire length of the roast-yard, and from which the ore is dumped directly upon the heaps.

These are made very large, about 40 by 80 feet, and some 6 feet high. Sound pine wood is used for the bed, and about one cord of wood is laid down for each 20 tons of ore in the heap. It is laid with great care, leaving air-channels to the center of the pile, while the surface of the wood is evened off with chips and smaller wood, and everything possible is done to obtain strict uniformity of combustion. After the main body of the heap is formed of the coarse ore, the ragging, or middling-sized ore, is added, being about a foot thick at the base of the pile, and tapering up to a few inches at the top. The fines are lastly added in the same way, though not so thickly at first, and a considerable additional supply of the latter is placed in heaps around the pile, in order to smother the fire after it is started.

The heap is fired at night, on account of the smoke, and when possible, it was found best to wait until several piles were ready and then fire them all at once, thus concentrating into one short period the harassing escape of fumes that would otherwise annoy the workmen who were building up or tearing down neighboring piles, just as much every time each single one was lighted.

A skilled and trusty man attends to the kindling and watching of the heap, and usually has to spend the entire night in working at it; filling up holes that appear from the burning away of the wood, and consequent sinking of the superincumbent ore; adding fines to those portions of the surface which appear too hot, and giving vent to those parts that show no sign of life. The entire success of the roast depends upon the building and lighting of the heap and its management during the first few days, and especially the first 24 hours.

And at this point, it may not be amiss to again call attention to the vital importance of having a proper and well-drained roast-ground. In the severe climate of Northern Ontario, it was sometimes necessary, in inaugurating a new plant, to build a heap on frozen ground, or ground that was not thoroughly drained. In both cases the results were miserable, the escaping steam seeming to completely impede the combustion, and the resulting heap, when torn down, revealing isolated spots, each containing many tons of ore that were not roasted at all, only the surface being slightly scorched, though the greatest pains were taken in building and managing the heap.

An absolutely dry and unfrozen ground is essential to success, and if snow falls, it must be carefully cleared away before laying the wood down, and after the wood is once in place, no snow or rain must fall upon either the fuel or ore. This can be protected by coarse canvas or boards, and well repays the labor and expense. I am not speaking of the slight rains and snows of more genial climates, but of the heavy Arctic storms of this northern country.

Where the ground is frozen, the results of the roasting are always extremely unsatisfactory. A few hours after lighting the heap, water begins to flow out from under it, and for a day or two, a continuous stream will pour out from the lower side of the pile, generating steam in quantities, and extinguishing the fire as soon as the lumps of ore are scorched a little on the outside.

The size to which the ore must be crushed is another most important factor in heap-roasting. Some ores, especially the ordinary iron pyrites or bisulphide of iron, can be dumped on the wood in lumps as large as a child's head without seriously affecting the results. Under the influence of the intense heat, these lumps crack asunder, and the fire penetrates into every crack and fissure, and eventually oxidizes every atom of the bisulphide. But with most monosulphide or pyrrhotite ores, the case is different, and the nickeliferous Sudbury pyrrhotite is perhaps the hardest of all ores to roast. Not the slightest swelling or fissuring of the lumps occurs, and the oxidation seems to penetrate only a certain short distance from the surface of the piece of ore. If the diameter of the lump is more than twice as great as the average depth of this penetrating zone of oxidation, there will always be found a central kernel entirely unchanged and unaffected by the roast.

It is essential, therefore, to set the crushers very close, 1¼ to 1¾ inches between jaws, and to keep them so, as the wear of the jaws will rapidly increase the size of the opening if they are not constantly attended to.

Owing to this necessity, a greater amount of the ore is crushed into ragging and fines than can be used to cover the heap in the ordinary manner. It is better to treat the surplus by itself than to run the risk of damaging the entire roast by covering the heap too closely.

The ragging may be roasted advantageously in a heap by itself, by dumping first a layer of coarse ore upon the wood, and then completing the heap with ragging, which should not, however, be much over two feet thick at any point. This low heap is covered lightly with fines, and will burn out in a couple of weeks. Very good results may thus be obtained.

But a greater difficulty is presented in dealing with the surplus of fines. I know of no better plan than to lay them down some 6 to 10 inches thick upon the ground where a fresh heap is to be built. The wood is then placed upon the fines, and the heap constructed as usual. After one or two heaps are burnt on this foundation, it will be found that the fines have become oxidized for perhaps half their depth from the surface. This top layer is then removed with the ore, and the fresh heap built upon the bottom layer until it, in its turn, is ready to remove, when a fresh lot of fines may be laid down and the operation recommenced. If all these points that have been described are carefully attended to, there will be no danger of too low a matte from the first smelting. At Sudbury, we nearly always add unroasted fine ore ("Green Fines") to the charge to prevent the formation of too rich a matte, with its accompanying evils.

But ordinarily the roasting is carelessly executed, and the smelting suffers accordingly. The most important factor in successful roasting is to always have sufficient ore on hand and being mined, so that none of the operations at the roast-yard have to be hurried. If, at any works, it is found that the furnaces are living from hand to mouth, that roast-heaps have to be broken into while still hot and burning, and that there is always a hurry for roasted material, it is time to stop all smelting operations and secure an abundant supply of roasted ore.

Nickel matte has many peculiarities of its own, and makes so many difficulties in handling, and in its property of welding to, and destroying, any iron surface that it touches, that it made us endless trouble at first till we learned how to handle it, no one before myself ever having tried to smelt nickel ores in such a furnace on a large scale.

About 7 tons of ore should need one ton of Connellsville, or equally good coke. Pine charcoal at 6 cents per bushel would probably be just about as cheap as coke at $8.00 per ton, but in the northern climate, where it would certainly get wet and damaged, which does not harm coke so much, the coke would be infinitely more advantageous. Hard-wood charcoal at the above price might possibly be cheaper than coke. It would be so near equal that only experiment on a large scale would determine the matter, but in any case, it would probably pay to use $\frac{1}{3}$ to $\frac{1}{2}$ coke, as the furnace is under much better control with the denser fuel.

TREATMENT OF THE NICKEL MATTE.

In nickel smelting, when the matte is obtained, it still remains to be refined, and only those who have been through such an experience realize the difficulties of disposing of it.

In the first place, it becomes a question of calculation whether it will pay better to ship the matte at about a grade of 25 per cent. as it is produced from the furnace, or to concentrate it on the spot by a second series of roasting and smelting operations. Until the local conditions, wages, scale on which operations are conducted, exact character of ore that is treated, etc., are known, this question cannot be answered. The matte is enriched by roasting it and resmelting it in a water-jacket or other furnace with quartzose flux to take up the iron. It is a question to be determined by circumstances whether the roasting should be executed in heaps, as with the ore, or whether it should be crushed and calcined in a few hours in calcining furnaces. Heap-roasting of matte takes about as long as the ore, because it has to be re-roasted 2 or 3 times, as it does not roast freely like the ore. But as there is only about one-sixth so much to handle as of the raw ore, the expense per ton of ore is not heavy. A matte of about 50 to 60 per cent. of nickel is produced by the so-called concentration smelting.

This concentrated nickel matte has a high point of fusion, and easily forms crusts and accretions.

It is impossible to smelt it in a furnace with brick fore-hearth, as may be advantageously done with the ore, for it soon fills up the front crucible, necessitating its substitution, and leaving a "salamander," weighing a ton or two, that is difficult to break up.

After much experimenting, I have returned to the old practice of using "steep," or a mixture of pulverized coke and clay for a fore-hearth, cutting in it a small crucible connected with the furnace-crucible by a deep groove. Out of this crucible, the rich nickel matte can be either tapped or ladled into molds, and as this method of procedure involves frequent, though very slight, repairs, it will save much delay to make the fore-hearth broad enough to permit of two such crucibles, side by side. Thus one can be repaired and dried while the other is in use.

The further treatment of the nickel matte, according to the old practice, is well known, and its description would be out of place in this connection.

Being expensive and slow, efforts are being made to improve upon it, and one of the principal nickel smelting companies at Sudbury is erecting a plant to Bessemerize this rich sulphide of copper and nickel.

According to the laws of chemical affinity, as modified by the high temperature employed, we know that the iron still remaining in the matte ought to oxidize first, forming, with silica, a slag that may be poured off. Next the nickel should oxidize and slag away, leaving behind the pure copper.

But whether such accurate results will be reached in practice seems to me somewhat doubtful.

In the Bessemerizing process, as applied to iron, the entire mass of metal remains homogeneous throughout the operation, the impurities being gradually oxidized, until it is all converted into steel. And the total amount of these impurities is only 4 or 5 per cent., so that the mass of fluid metal operated upon is not perceptibly lessened.

But in Bessemerizing a mixture of the sulphides of iron, copper, and nickel, the number of different chemical compounds, having differing specific gravities and tending each to form its separate stratum in the converter, is too great to even enumerate.

As soon as sufficient sulphur is removed to correspond to the iron present, we shall have a layer of oxide of iron (combined with silica from the converter lining) on top, while below, the sulphides of nickel and copper will remain comparatively unaltered. Then may come a period, when we have the same silicate of iron on top, followed by a little silicate or oxide of nickel, whilst some metallic nickel has formed and sunk to the bottom, and the rest of the nickel, in its original condition of sulphide, forms a stratum below the unaltered sulphide of copper.

These reactions and products increase in number and complexity as the operation advances, and remembering the great difficulties encountered in Bessemerizing even so simple a substance as copper matte, one cannot help feeling some curiosity as to the practical success of this operation.

That nickel and copper can be rapidly reduced from the condition of a matte to that of separated metals, the author has convinced himself. But business considerations prevent the further elucidation of this subject.

The final treatment of the nickel-copper alloy, or of the already separated metals, does not fall within the scope of this work.

But it must be evident to every one familiar with the facts, that the commercial electrolysis of copper on the one hand, and the electrolytic deposition of nickel in our nickel-plating establishments, on the other hand, point out a path to follow that is too plain to be neglected.

And as our chemists find no difficulty in precipitating, with the electric current, chemically pure copper from a solution containing both copper and nickel, and then, by slightly altering the conditions, precipitating all the nickel in absolute purity, from the same solution, and with the same current, it would seem that our refiners might reasonably expect to effect the same results on a commercial scale, especially as there is practically no loss of acid in the operation.

Nor can I see any reason why nearly all our metallic nickel should be offered to the trade in little cubes less than an inch square. Of course, this peculiar form has resulted from the practice of the nickel refiners to reduce the oxide of nickel obtained by the methods now in use to metallic nickel. Being mixed with rye meal, as a reducing agent, it is formed into these little cubes and a number of these packed in crucibles are exposed to a sufficient heat to reduce the nickel to a metal without fusing it.

This makes a small, porous fragment of metal, suitable for solution in acids, and where nickel is to be used in minute quantities.

But it adds materially to the expense of refining, and there is really no more reason why nickel should be so treated than copper or iron.

Although the fusion point of nickel is rather high, yet a sufficient temperature to make nickel pour as readily as copper is obtained without difficulty in metallurgical practice, and there is little doubt that before long nickel will be refined in bulk and cast into suitable ingots, as is copper or lead.

Indeed, at Vivian & Company's nickel works in England, a small reverberatory, heated by gas, has been in use for several years for refining nickel, some 2,000 pounds being refined at a charge; and the superb display of solid nickel articles and ingots, made by Joseph Wharton of Philadelphia, shows that he experiences no difficulty in melting and casting nickel like other metals.

CHAPTER XIV.

REVERBERATORY FURNACES.

This method of smelting copper ores is peculiarly English, the reverberatory furnace having practically had its origin in Swansea, where it has, during the past two centuries, undergone various changes and improvements, by which its capacity and economy have been considerably increased without any radical alteration in its original form or practice.

The American reverberatories are modeled closely after their English prototypes, and present no new features worthy of note, a constant tendency toward increased size and capacity being almost the only point in which any difference can be detected.

This particular branch of metallurgy having engaged the attention of English and American authors to a greater extent than any other, nothing would be gained by a mere repetition of what may be found in the modern text-books, and the writer prefers to devote his own work to those practical details of construction and management that are yet wanting, and which he hopes may supplement the more strictly scientific information just referred to.

While the blast-furnace has replaced the reverberatory to a considerable extent in the United States for ore-smelting, the latter is still generally preferred for matte concentration, and especially in its last stage, or the production of "blister copper."

The processes executed in the reverberatory furnace in this country may be divided into the following classes, each of which demands separate consideration:

1. Ore smelting.
2. Matte concentration.
    - *a.* By fusion of calcined matte.
    - *b.* By an oxidizing smelting.
3. Production of blister copper.
4. Copper refining.

As the first three of these processes are carried out in exactly the same type of furnace, their more detailed description may well be preceded by some consideration of the construction of a reverberatory furnace.

The excavation for the foundations should be 18 inches in every direction larger than the proposed furnace, allowance being made for the space occupied by the stack or down-take at one of the front corners. A depth of 4 feet from the floor level is sufficient, and a permanent drain should keep the pit free of water. Exceptional circumstances may require a greater depth of so much of the excavation as corresponds to the foundation of the stack. Two longitudinal walls are now laid in such a manner that a 4-foot space is left under the main body of the furnace, extending from the back of the ash-pit to a point directly under the future front wall of the furnace.

This is arched over with two 4-inch courses of red brick, upon which come one or two 4½-inch courses of fire-brick. The bridge wall and two lateral walls of the ash-pit are also begun from the same level. It is also well to carry up the side and front walls of the furnace from the very bottom, using red brick for all underground work, and filling the space between and outside of the walls with stone or slag, broken *in situ* with spalling-hammers, and firmly united with liquid mortar, or by pouring in the pots of slag as they come from the blast-furnace.

It is quite customary to fasten the looped tie-rods for the perpendicular buckstaves by merely bending a hook at the end of the rod and building it into the wall, trusting to the weight of the superincumbent mason-work to prevent their drawing out. It is a much safer plan to introduce the tie-rods at a lower level, giving them sufficient length and inclination to pierce one of the central longitudinal walls, and providing each with an eye through which passes a long continuous bar of iron, which thus firmly holds all the tie-rods belonging to one side of the main body of the furnace. This bar is fastened to its fellow of the other side by a few short cross-rods, and the lower set of loops is thus firmly held in place, and far below any chance of being melted in two—an accident that would certainly occur in the course of time if they crossed the entire furnace *above* the subterranean arch.

The inclosing walls of the furnace having been built to within a foot of the floor surface, the hearth proper of the furnace is laid in the shape of an inverted arch, its lowest point in the center being in contact with the upper convex surface of the 4-foot subterranean arch, while its sides rise at the rate of about half an inch to the foot. It is also slightly arched longitudinally, and should be well keyed and grouted, as it is intended to be so constructed as to prevent the possibility of its being floated up by any breaking through of the molten contents of the sand-earth.

The hearth is now inclosed by side-walls of fire-brick, 4½ inches thick, which support the arch when the proper height is reached. These are incased by strengthening walls of red brick, while they are protected on the inside by a 9-inch lining of fire-brick, which can thus be renewed when necessary without interfering with the arch.

As the hearth is an elongated oval, narrowing to about 18 inches at the skimming-door, while the external shape is usually that of a rectangle, it follows that the four exterior corners consist of useless pillars of rubble or brick-work. These are sometimes avoided by conforming the external shape to that of the hearth, and inclosing the fire-brick with thin wrought-iron plates, or thicker plates of cast iron, often perforated with rows of holes to diminish their weight. The latter arrangement causes some difficulty in placing the buckstaves, and presents no decided advantage over the older plan of inclosing the hearth in a rectangular mass of brick-work, only 4 inches thick at the widest portion of the hearth, and increasing rapidly toward each extremity. Two heavy vertical cast-iron plates support the hearth at each end—the "conker-plate" giving strength to the bridge-wall, while the front plate is placed just below the front door, the narrow horizontal skimming-plate resting upon it, and determining the eventual thickness of the sand-bottom.

The bridge-wall is a massive structure of fire-brick, perforated by an air-passage about 3 inches by 10 inches, which has the conker-plate for its anterior wall, while a lighter casting forms its posterior boundary. A large blow-hole, or opening for the admission of air, should be left on each side of the furnace in the angle formed by the posterior wall of the main portion of the structure and the wall of the fire-box. These orifices are used only when an oxidizing atmosphere is desired, as in the concentration of matte, the making of blister copper, etc., and can be tightly closed with clay under ordinary circumstances.

The fire-box is inclosed with a 9-inch wall of fire-brick, which may be strengthened by a casing of common brick, if desired.

Where coal is used for fuel, particular attention should be given to placing the grating-holes (that is, the orifices in the sides of the fire-box just above the grate, through which bars are introduced to cut away the clinkers) in a convenient position.

The arch is best constructed of "Dinas" or silica brick, which last much longer than ordinary fire-brick, and should have a rise of an inch to the foot, pitching downward quite abruptly from a point slightly anterior to the bridge-wall, until it approaches to within twelve or fourteen inches of the skimming-plate at the front door. Its shape, as well as the size and proportions of the space between bridge and roof, has much to do with the heating qualities of the furnace, and must vary with the character of the fuel and with other local conditions. The extreme front row of arch bricks, forming the posterior wall of the flue opening in the roof, is called the "vulcatory," and from its situation is so exposed to wear and heat as to require frequent renewal.

The flue opening itself is of a trapezodial form, being inclosed laterally between the two converging walls of the hearth, while it has the vulcatory for its posterior and the front wall of the furnace for its anterior boundary.

Its size and proportions are matters of paramount importance, as the heating capacity of the furnace as well as its consumption of fuel depends principally upon it and upon the size and shape of the flue proper, that is, the canal connecting the hearth with the chimney.

No precise rules can be laid down in this matter for the guidance of the inexperienced, as each individual case must be judged upon its own merits until constant experimenting has determined the question.

The uncertainty and difficulty pertaining to this matter may be best appreciated when it is known that, of half a dozen furnaces in the same building, constructed from the same plan and apparently identical in every particular, fed with the same fuel, and smelting the same ore, no two behave in the same manner, and therefore each must have the size and shape of its flue suited to its needs. In general terms, it may be stated that a large flue will cause a greater consumption of fuel and a quicker heat, unless a certain limit is over-stepped, beyond which the fuel will be burned without a corresponding rise of temperature. It is quite obvious, therefore, that the economical smelter will seek to throttle his flue to the greatest possible extent compatible with the rapid production of the required temperature. The flue should be narrowest at its junction with the furnace, and expand considerably as it enters the stack, having at least 50 per cent. greater area at the latter point than at the former. Its size is altered by introducing or removing a little dam of sand at the end nearest the furnace, one of the slabs with which it is covered being removed for that purpose. When experiments of this nature are executed to determine the most advantageous flue area, it is important that the change in size should be sufficient to produce some plainly marked effect either for the better or the worse; otherwise, it is a mere groping in the dark. The weather, force or direction of the wind, and general condition of the atmosphere may often produce an impression sufficiently powerful to entirely mask the changes brought about by the alteration of the flue area, so that a considerable period may be necessary to properly estimate the good or evil resulting from the efforts of the smelter.*

The buckstaves supporting the furnace should be of wrought-iron, and sufficiently strong and numerous effectually to prevent any spreading of the sides or arch.

This is especially important with blister or refining-furnaces, where the weight of the molten bath may amount to 10 or 15 tons, which, combined with the lateral pressure caused by the high temperature, produces an expansive force that is almost incredible.

The tie-rods should be of 1¼-inch square iron, provided at the extremities with loops, and not with thread and nut.

Where wood is used as a fuel, a row of small openings should be left in the arch over the anterior edge of the fire-bridge; an arrangement that insures the combustion of the gases in the hearth where they are needed, instead of in the chimney, where they may produce a most detrimental effect, destroying the fire-brick lining within a few days.

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* The proportions of numerous reverberatory furnaces are given in Egleston's monograph on *Copper Refining in the United States*, as well as in a valuable paper by H. M. Howe, E. M., in the government report edited by Albert Williams, Jr.

Even with this precaution, it is sometimes necessary to leave an opening near the base of the stack, to prevent excessive flaming just after adding fresh fuel.

A damper playing in a hinge, and fastened to a cast-iron frame, should invariably be placed on the summit of the stack. By this means, the draught can be effectually regulated, or entirely cut off if desired—as when charging fine, dry ore—without resorting to the familiar but slovenly practice of removing a slab from the flue and inserting an iron plate.

The capacity of a reverberatory furnace and that quality of rapid, fierce heating so essential to economical ore smelting, are largely dependent upon the proportions of the flue and stack, and while the former may easily be made too large for economical work, the latter is oftener too small for the sharp draught required.

These remarks apply especially to reverberatories used for smelting ore, where the object is to attain the highest possible temperature in the shortest time. Blister furnaces, or those devoted to matte concentration by the old method of "sweating down," do not require such a powerful draught, as the processes are slow, and the temperature required comparatively low. For this reason, they may be provided with down-takes and flues entering a stack common to several furnaces; while each ore-smelting reverberatory should have its own independent stack, without either flue or dust-chambers; any loss in flue-dust that may occur after closing the damper while charging being fully counterbalanced by the saving in time and fuel, except under peculiar circumstances.

The fire-brick lining should be entirely independent of its surrounding walls, so that it can be easily removed and renewed, and a 2-inch air-space should be left between the same, connecting with openings near the base, so that a current of cool air may constantly surround the heated lining.

As has been already mentioned, that portion of the stack below the entrance of the lateral flue should be left entirely empty nearly to the ground level. in order that an elastic cushion may be provided for the flame as it enters the stack from the furnace.

The height of the stack need seldom exceed 60 feet, unless local conditions affect the draught. The immediate proximity of higher roofs or abrupt hills often injures the draught to a most serious extent, which circumstance should always be borne in mind in planning new works.

From a stack 65 feet high, the writer has removed 15 feet, and subsequently added 30 feet, without affecting the working of the furnace in the slightest degree; but the height just mentioned may be regarded as safe under ordinary circumstances.

The position of the large doors and other openings of the building should be so arranged that under no conditions can the wind blow across the ash-pit of the furnace in such a manner as to counteract the draught. It is not uncommon to find furnaces that, during certain winds and other atmospheric influences, fall off to a very marked degree in their duty.

There should be an ample free space about a reverberatory furnace; at least 15 feet on the tapping side, and the same distance in front, while a space of 12 feet on the charging-door side will suffice. This should be well drained and paved with brick on edge, or with cast-iron plates.

The arch being completed and the wooden pattern removed, the furnace is taken in charge by the smelter, who, with the blacksmith's aid, proceeds to the proper tightening of the tie-rods; the side buckstaves having been already sufficiently drawn up to keep the arch in place. This process has been described in the chapter on Calcining Furnaces, and pesenets no peculiarities. The empty hearth should be covered with a 2-inch layer of fire-clay, to prevent adhesion of any metal that may possibly make its way through the sand bottoms.

A small fire may be at once built on the surface of the clay stratum and in the ash-pit, and should be maintained for at least four days, slightly raising the temperature, until, at the expiration of this time, a dark-red heat is attained, and the cessation of aqueous vapors from the side walls and subterranean arch shows that every particle of moisture is removed.

The grate-bars are now placed in position—being mere rods of inch-square wrought-iron—and the fire, being shifted to its proper position, is gradually urged for twelve hours or more, until the whole interior is of a light-red heat.

Then, and not until then, should the material for the smelting-hearth be introduced.

This consists essentially of silica, and may be in the shape of well-washed beach sand, or crushed sandstone, or of pulverized quartz, first roasted in lumps and quenched while hot, to impart a high degree of brittleness and greatly facilitate its crushing.

A beach sand employed for this purpose in Swansea, and analyzed by Percy, had the following composition:

	Per cent.		Per cent.
Silica	87·87	Magnesia	0·21
Alumina	2·13	Carbonic acid and water	2·60
Sesquioxide of iron	2·72	Total	99·32
Lime	3·79		

This is not so refractory as the crushed sandstone employed by some of the Eastern American smelting-works, or the pulverized quartz used for reverberatory bottoms in Butte, Montana, which, according to the author's tests, contain respectively, when dry, 95·3 per cent. and 97·2 per cent. of insoluble residue, presumably silica.

Two methods are pursued in making reverberatory hearths. Either the sand chosen contains enough bases to be slightly fusible, or a small proportion of crushed slag or other similar substance is added so that the sand may become slightly agglomerated by the intense heat to which it is subjected; or secondly, the material selected is practically infusible, and the cementation of its particles is effected by smelting small and repeated charges of fusible material upon the slightly hardened surface of the same, until it is solidified into a hard and impermeable mass.

The author prefers a combination of these two systems, using the first method for the lower hearth, and the second for the upper or true hearth; as it is usual to put in two separate hearths, the upper one being comparatively thin, so that it can be easily removed when worn out.

The total height from the floor of the furnace to the upper surface of the skim-plate being perhaps 30 inches, the lower hearth (including the clay bottom) should have a thickness of 18 inches and the upper of 12 inches, both of them being somewhat concave in shape, so that a basin is formrd some 5 inches deeper than the skim-plate in the center, and sloping from every direction toward the tap-hole.

The size of the hearth material is a matter of less importance than is often supposed, provided that, if at all coarse, sufficient fine dust is present to fill all interstices and prevent porosity.

If good crushing facilities can be had, it is well to pass everything through a 16-mesh screen, but the author has used even a 5-mesh without evil results.

This, of course, refers to sandstone or quartz rock. Natural sand usually requires no sizing process, unless mixed with gravel.

The utmost care should be taken to prevent the introduction of any foreign material, especially of an organic nature, as the gases generated therefrom may easily cause a ruinous flaw or blister in an otherwise perfect hearth.

Such unfortunate results, however, are usually counteracted by the thorough calcination that all sand must undergo previous to the final smelting.

A sufficient amount of the sand—usually from 4 to 5 tons—being thrown into the heated furnace (either as such, or mixed with from 3 to 5 per cent. of pulverized slag), a moderate fire is maintained, while a steady stirring and rabbling is kept up through the side and front doors, until every particle of moisture and carbonic acid and other gases is expelled, and the heat gradually raised to such a temperature as to insure the decomposition of all organic material. This operation may require from 3 to 8 hours, according to the nature of the material. Both the temperature and time are matters of great importance, having a marked effect upon the final result; but can only be learned by experience, as they vary with each different sand.

Toward the close of this period, the sand is gradually brought into the proper shape for the bottom, and thoroughly pressed and stamped into place by means of long paddles and stampers, worked through the door openings. No great pains need be expended upon the lower hearth, as it will, of course, be entirely covered and its shape obliterated by the superior layer.

The doors are then closed; the tap-hole bricked up and covered with a heap of sand, and every crack and orifice about the whole furnace completely stopped and closed. The fire is gradually urged until the highest possible temperature is reached and maintained for a couple of hours, the entire period of heating requiring from 6 to 14 hours, according to the heating capacity of furnace and fuel. The interior condition of affairs is watched through a peep-hole in the front door, which is provided with a clay plug. After a proper maintenance of the highest temperature, the fire is gradually slackened and the furnace cooled down. This operation demands the greatest care and circumspection, as the premature opening of a door or a sudden draught of cold air may cause the appearance of a crack or blister in the porcelain-like surface of the sand hearth. Several hours must elapse before the doors can be taken down and the results of the operation inspected. The interior of a reverberatory furnace under these circumstances is quite an interesting sight.

Long stalactites of molten fire-brick hang down from the arch over its entire surface. The side walls are not only glazed, but actually fused until they begin to soften to a considerable depth, and the hard and glistening semi-fused surface of the new hearth is strewn with fragments of brick from the crown, and little heaps of molten fire-clay corresponding to the pendent stalactites.

Unless very serious cracks exist, no notice need be taken of them, and blisters and irregularities may be entirely overlooked, as the upper hearth is to bear the brunt of the work. After slow and perfectly even cooling to a dark-red heat, about 1,800 pounds of moderately basic, fusible slag, crushed to the size of chestnuts, are spread over the entire surface, being charged by means of long-handled paddles, and on no account thrown in carelessly, to be subsequently leveled with rabbles, as is often done. The doors being again tightly closed, the slag-charge is quickly smelted down, two hours being amply sufficient for this purpose. This layer of slag will be entirely absorbed by the porous sand bottom, which, after a second cautious cooling, should be again charged with a somewhat larger burden of slag, with which are mixed a few hundred pounds of low-grade matte (30 per cent.). After this is melted down, a considerable portion will probably be found in a pool near the tap-hole, from which it should be immediately evacuated. If the furnace is to be used for concentration work, or especially for the production of blister copper, still another charge should be melted on the lower hearth, consisting principally of matte of the same grade as the former, and should be tapped as soon as sufficiently liquid. In this way, the lower hearth will be pretty thoroughly saturated with matte of low tenor, thus preventing the absorption of an equivalent quantity of richer metal in case the same should penetrate from the upper hearth. The tying up of a large amount of copper may thus be guarded against.

After a final cooling, the sand to form the upper hearth is thrown in, and, after a careful calcining and leveling, should be stamped into place with the utmost care, sloping gently toward the tap-hole from every point.

The fusion is executed in the manner already described, and in addition a careful watch should be maintained at the peep-hole to observe and remove any fragments of brick that may fall from the arch during the early period of the fusion. After the softening of the sand has once begun, no further manipulation is permissible.

The cooling after the fusion must be executed with extreme care, to prevent cracking and blistering, and as soon as a dull-red heat is reached, the slag and matte charges already enumerated should be successively melted, with alternate periods of cooling.

As a final preparation before introducing the first ore-charge, a small quantity of finely crushed slag should be thrown around the entire edge of the hearth at the junction of sand and fire-brick.

Above this, a thick bolster of "fettling" (mixed fire-clay and crushed quartz) should be tightly forced into the angle between hearth and side-walls, including the bridge-wall.

An hour's brisk heat will dry and consolidate the fettling, and the regular work of the furnace may begin—small charges being used at first, and no large quantity of metal allowed to accumulate before tapping.

MANAGEMENT OF FURNACE.

The experienced furnace-man constantly watches his furnace with reference to the safety and condition of its bottom. After a few hours' firing on a fresh charge, the workman introduces his rabble, and by the feeling of the sand when gliding over the bottom, determines at once the condition of things.

If slippery and sticky, it indicates that portions of the charge still adhere to the hearth. These are removed as far as possible by the rabble, and dissolved by a short additional heat, until the rabble glides smoothly over a plane, granular bottom, which is the upper surface of the hearth proper. After this condition is once attained, every additional moment of high temperature is not only wasted, but is positively detrimental to the hearth, which lacks the protection of the semi-fused ore, the liquid matte soon attaining a high temperature, and, if exposed to the air, boiling in a manner that may prove highly dangerous to the bottom. This is the case when concentrating matte or making blister copper—operations very severe on the bottom, but rendered less dangerous by being conducted at a lower temperature than is required for smelting proper.

Equally detrimental may be a high temperature with a small charge, where the unprotected portions of the bottom may become so softened as to rise in large flakes, being literally floated up by the superincumbent metal.

Any large piece of iron, such as a rabble-head, may cause a hole in the bottom, and in endeavoring to float up an old bottom, nothing is more effective than the introduction of a number of large fragments of old iron. A bottom may be often patched to advantage when only locally damaged. When any such condition is discovered, the hearth should be immediately emptied, and the damaged portion, which usually shows a decided cavity or depression, should be most carefully emptied, fresh sand being repeatedly introduced and again removed with the rabble until it is completely dry. The hole should then be filled and leveled up with ordinary bottom sand, which must be fused and saturated with the same precautions as in the case of the original bottom. In this way, a bottom may often be saved for many months at a very slight expense.

In direct connection with the management of the bottom is the proper fettling of the furnace. The entire life of the side walls and safety of the bottom depend upon the care and conscientiousness observed in maintaining the dam that incloses the molten, liquid pool and protects the fire-brick. In default of this safeguard, the side walls are quickly undermined, a groove several inches in depth being cut into the mason-work during the smelting of a single basic charge. Nothing then remains to prevent the descent of the metal between the wall and bottom until the latter is floated up and ruined, and a large amount of copper temporarily lost. The best fettling is formed of pure, white quartz, crushed through a 3-mesh screen and mixed with sufficient plastic fire-clay to form balls, which may be placed at exactly the required point, and forcibly pressed and molded into place. The quartz may be replaced by ordinary bottom sand, which, however, is less permanent and solid. When smelting basic ore, the hearth may require fettling after every charge; but with a quartzose mixture, days may elapse without any necessity for renewal. Safety in this particular is only obtained at the expense of constant watchfulness.

The size of the fire-box and depth of grate below the upper surface of the bridge are very variable factors, depending upon the quality of the fuel and degree of temperature.

The best and most economical results are obtained by the use of the clinker grate, which is virtually a gas generator, a deep layer of clinkers being maintained upon the grate-bars, penetrated by numerous openings through which the air passes, being heated to a high temperature before it unites with the gas generated from the coal, which lies upon the upper surface of the bed. A certain proportion—from one-third to one-half—of caking coal is required for this method of combustion, and the grate-bars must be at a much greater depth than for ordinary non-caking fuel.

Lignites, or any free-burning, non-caking coal, require a shallow grate and a large flue, while wood behaves in much the same manner, requiring, however, the introduction of air through holes in the roof above the bridge, on account of the great volume of combustible gases generated.

It is almost impossible to give any general rule for the amount of fuel required for a reverberatory furnace. When engaged in smelting ores, a much larger quantity must be consumed than in making blister copper or in the refining process, where only a very moderate temperature is needed for a considerable portion of the time. The following table gives the average results obtained by the writer, and comprises several varieties of fuel, and most of the different operations executed in the reverberatory furnace:

TABLE OF REVERBERATORY WORK.

OPERATION.	Fuel.	Size of fire-box in feet.	Size of hearth.	Fuel used per twenty-four hours.
Ore smelting...	Lignite.	3¼ × 4	9 × 14½	5¼ tons
Same furnace..	Bituminous coal.	3¼ × 4	9 × 14½	3¾ tons
Same furnace..	Wood.	3¼ × 4	9 × 14½	6¼ cords
Ore smelting...	Bituminous coal.	3½ × 4	10 × 15	3⅞ tons
Blister smelting..........	Bituminous coal.	3½ × 4	10 × 15	2¼ tons
Blister smelting..........	Wood.	3½ × 4	10 × 15	4¾ cords
Refining.......	Wood.	3¾ × 4½	9 × 13½	5 "
Refining.......	Coal.	3½ × 4	9½ × 14	3 tons

One ton = 2,000 pounds.

Anthracite may be employed in this form of furnace, when provided with a tight fire-box and an artificial blast of considerable volume and slight pressure.

ESTIMATE OF LABOR AND MATERIALS FOR BUILDING AN ORDINARY REVERBERATORY FURNACE.

Size of hearth inside, 9½ by 14 feet. Besides the independent 9-in lining of fire-brick, inclosing the hearth, there is a backing of 4½ inches of fire-brick, from which the arch takes its spring, and a casing of red brick, 12 inches thick at the widest part of the hearth, and rapidly increasing in thickness toward either end. The fire-box is 3½ by 4 feet, and the bridge 30 inches across.

All these, as well as the remaining proportions, may be altered within ordinary limits, without materially affecting this estimate.

Detailed drawings of the best modern reverberatories will be found in a succeeding section.

EXCAVATION FOR FOUNDATIONS.

This should be about 18 inches larger in every direction than the proposed stone foundations, and about 3 feet 9 inches deep ordinarily, while the excavation for the stack-foundation is put at 6 feet deep. The amount of earth usually removed is about 1,940 cubic feet, costing, to dig and remove, about 2½ cents per cubic foot. Total, $48.50.

STONE WORK.

The stone foundation walls are usually continued to within about 8 inches of the surface of the ground, and will require about 700 cubic feet of stone work, which includes the stack-foundation complete at 35 cents per cubic feet. Total, $245.00.

BRICK WORK.

The fire-brick required will be:	Cubic feet.
For lowest fire-brick arch over four-foot vault	25
Main concave arch, forming bottom	114
Front wall at skimming-door	6
Flue and covering slabs	22
Bridge-wall, estimated as if solid	35
Two side walls of fire-box	110
Rear wall of fire-box	15
Hearth lining	100
Lining of stack, estimated at 14 inches thick for lower half and 4½ inches for upper half, stack 60 feet high, and assumed to be 32 inches square in the clear throughout, to simplify calculation	690
Total cubic feet	1,117

Assuming 18 fire-bricks to the cubic foot, we have a total of square fire-brick of 20,106. To these must be added the following so-called "shaped brick," to avoid cutting in turning arches, door-jambs, etc., and in laying skew-backs.

	No. of brick.
Bull-heads	250
Side skew-backs	250
Jamb-brick	60
Wedge brick	30
Soaps	80
Splits	80
Total	750
Added to the square brick = grand total	20,856
Say 21,000 at $40 per thousand	$840.00
The main arch of the furnace will require 2,000 Dinas brick at $60 per thousand	$120.00

FIRE-CLAY.

For laying the above brick will be required:	
Raw fire-clay, 5 tons at $8	$40.00
Burnt fire-clay, 5 tons at $8	40.00
	$80.00

ORDINARY CLAY.

10 loads at $2	$20.00

The red brick required are:

	Cubic feet.
For side walls of vault under furnace	192
For double 4-inch arches over vault	43
Front wall under skim-door	84
Side casing walls and solid corners	408
Side casing walls of fire-box and ash-pit	128
	855
Casing of 50-foot stack, assuming it to average 12 inches thick for the entire distance	1,260
Grand total	2,115
Assuming 25 brick per cubic foot—to allow for waste, we have total number of red brick, 52,875—say 53,000 at $8	$424.00

LIME, SAND, AND CEMENT.

To lay the above brick requires:

50 barrels lime at $1	$50.00
12 barrels cement at $1.50	18.00
25 loads sand	36.50
Total	$104.50

IRON WORK.

Assuming that no proper expense is to be spared, the following iron work is required, being somewhat heavier than is commonly used, though no more than is needed for a strong and permanent furnace. Most of these plates are ribbed to increase their strength.

	Pounds.
Bridge plate, 5 feet long, 32 inches high, 3 inches thick, tapered and cored with holes to lessen weight	1,120
Front plate (across entire front and below skimming-door), 6 feet 3 inches long, 2 feet high, 3 inches thick, tapered and perforated	1,110
Ribbed plate to support rear wall of fire-box, 5 feet long, 9 inches wide, 1 inch thick, with heavy rib on top	182
Ribbed plate to support the fire-box portion of bridge-wall, 5 feet long, 9 inches wide, 1½ inches thick	246
Rear bridge-plate, which forms rear wall of air channel through bridge, 5 feet long, 2 feet 8 inches high, and 2 inches thick (tapering)	500
Plate to support one end of fire-box and contains frame for sliding door, 32 inches wide, 36 inches high, 1 inch thick	200
Carried forward	3,358

Brought forward	3,358
Skimming-block, forming threshold of skimming-door, and easily removable, 26 inches long, 8 inches wide on top and 5 inches below, and 10 inches high	430
Plate to protect brick-work at side-door with opening for door, 6 feet 9 inches long, 5 feet high, $\frac{3}{4}$ inch thick	725
4 skew-back plates on main furnace, above and below, each 17 feet long, 4 inches wide, 1 inch thick, with heavy rib	1,061
Plate to support flue, 3 feet long, 24 inches wide, 1 inch thick, with light ribs	246
Chimney cap and damper	549
Frames for fire-door and charging-door	181
2 bearing bars for grates and 1 grating bar, 5 feet long and 2 by 3 inches wide	305
Total cast-iron for reverberatory furnace	6,855

WROUGHT-IRON.

	Feet.	Pounds.
Tie-rods of $1\frac{1}{4}$ inches round iron.		
9 upper and 9 lower across main part of furnace	252	
3 upper across fire-box	24	
2 upper longest	45	
2 upper longitudinal on main furnace	34	
2 upper uniting in ring	16	
7 lower hooks in fire-box	18	
2 lower hooks at end of fire-box	12	
2 lower hooks at end of main furnace corner	10	
6 lower hooks in front	30	
52 loops for above to go over buckstaves	156	
30 loops to connect irons across vault	90	
	687 =	2,900
Grate-bars, 24 of inch square, 4 feet 3 inches		345
Arcs, levers, and chains for fire-door and charging-door		92
Skimming-bar, 6 feet long, $\frac{5}{8}$ inch by 2 inches		26
Bolts and nuts about furnace		72
Clamps for flue-slabs and brick doors		54
Chain to damper		14
Total, except old rails		3.503
Buckstaves, old rails, 90 pounds to the yard.		
9 for each side of main furnace, at $6\frac{1}{2}$ feet	117	
5 for both sides of fire-box, at $6\frac{1}{2}$ feet	33	
2 long at rear end of fire-box, at 9 feet	18	
2 at main end corners, at 6 feet	12	
6 at front of furnace, at 5 feet	30	
2 light ones for fire-door	6	
2 longitudinal rails under vault to connect irons from each side, at 16 feet	32	
	248 =	7,440

WROUGHT-IRON FOR CORNERS OF STACK.

480 feet of ½ inch by ¾ inch iron for uprights	605
360 feet ¼ inch by 1¾ inch iron for cross straps	532
Total	1,137

RÉSUMÉ OF IRON WORK.

Cast-iron, 6,855 pounds at 2½ cents	$171.36
Wrought-iron, 4,640 pounds at 2 cents	92.80
Old rails	56.04
	$320.20

LUMBER.

Lumber for main furnace arch, foundation arch, and chimney scaffolding, 2,360 feet at $18	$42.48

LABOR ON REVERBERATORY, EXCEPTING FOUNDATIONS.

Mason's labor, 160 days at $4.00	$640.00
Helper's labor, 165 days at $2.00	330.00
Carpenter's labor, 9 days at $3.00	27.00
Blacksmith and helper's labor, 8½ days at $5.00	42.50
Ordinary labor, 35 days at $1.50	52.50
Grading and clearing up	66.00
Superintendence	124.00
Incidentals	37.00
Total labor	$1,319.00

SUMMARY OF TOTALS.

Excavation	$48.50
Stone-work	245.00
Fire-brick	840.00
Dinas brick	120.00
Fire-clay	80.00
Ordinary clay	20.00
Red brick	424.00
Lime, sand, and cement	104.50
Iron work	320.20
Lumber	42.48
Labor, superintendence, etc	1,319.00
Grand total	$3,563.68

TOOLS.

The tools for a reverberatory smelting furnace should embrace:

1 long and 1 short paddle.
4 ordinary skimming rabbles.
6 ordinary stirring rabbles.

2 long stirring rabbles.
2 long clay stampers (for repairing).
2 short clay stampers (for repairing).
4 steel tapping-bars.
1 striking hammer.
1 sledge-hammer.
4 steel grating-bars, assorted lengths.
1 coal shovel.
Ordinary shovels.
1 rake.
1 iron wheelbarrow.
1 slag barrow.
Various hooks, pokers, etc.

Weighing in the aggregate (excepting barrows, shovels, and such tools), iron, about 1,300 pounds; steel, about 275 pounds.

All rabbles and bars are made and repaired at the furnace, the heads of the former being usually imported from England, and welded to the wrought-iron bar.

ESTIMATE OF COST OF RUNNING A REVERBERATORY FURNACE.

It is assumed that coal, ore, and fettling materials are of good average quality, and that the cost of superintendence and similar expenses is divided among six furnaces. The helpers of neighboring furnaces assist each other in charging and removing slag.

The allowance for repairs, as well as all the figures given, may be taken as reliable under the conditions assumed, as they are the average results of a year's actual work.

The cost of preparing and smelting in the two sand bottoms is also given, the figures adopted being the average of nine such operations, and consequently including such mishaps as will occasionally occur.

The general expenses of the works are too variable to be properly considered in such an estimate. The same may be said regarding fluxes.

COST OF REVERBERATORY FURNACE BOTTOM.

A—Lower Bottom.

Fire-sand (or prepared quartz), 4·75 tons at $8.......	$38.00
Coal for preliminary heating and fusion, 6·5 tons at $5	32.50
Two smelters at $3 (24 hours' operation).............	6.00
Two helpers at $2................................	4.00
Carried forward.............................	$80.50

Brought forward	$80.50
One laborer	1.50
Proportion of coal transportation	.25
Proportion of 2 foremen at $4 each	1.33
Lights, oil, soap, etc	.65
Repairs on furnace tools	.60
Clay and sand for doors, tap-hole, etc	.15
Total	$84.98

B—Upper Bottom.

Fire-sand (or prepared quartz), 3·9 tons at $8	$31.20
Coal for sand-roasting and fusion, 5·2 tons at $5	26.00
Two smelters at $3	6.00
Two helpers at $2	4.00
Two laborers at $1.50	3.00
Transportation of slag and matte to soak bottom	.85
Proportion of coal transportation	.20
Proportion of 2 foremen at $4 each	1.33
Clay and sand for final fettling, doors, tap-hole, etc	1.60
Lights, oil, soap, etc	.65
Repairs on furnace tools	.75
Total	$75.58
Grand total	$160.56

COST OF RUNNING REVERBERATORY SMELTING FURNACE 24 HOURS.

Coal, 4·3 tons bituminous coal at $5	$21.50
Clay and sand for fettling, 220 pounds	.45
Cheap clay and loam for luting; sand for matte beds, 800 pounds	.60
Proportion of renewing upper bottom annually	.24
Proportion of renewing main arch annually	.69
Proportion of renewing flue quarterly	.21
Proportion of other repairs of furnace	.88
For renewing tools and barrows	1.36
Repairs on tools and barrows	1.25
Lights, oil, soap, chalk, etc	.72
Refuse wood for drying matte beds (only happens exceptionally)	.12
Repairs on furnace doors	.13
Two smelters at $3	6.00
Two helpers at $2	4.00
One laborer (on day shift only)	1.50
Proportion of coal transportation	.25
Proportion of ore transportation	.40
Proportion of selecting slag and building dump	.25
Proportion of removing ashes	.25
Proportion of 2 foremen (day and night) at $4 each	1.33
Allowance for extra help (exceptionally)	.33
	$42.46

As such a furnace should easily smelt 12 tons per twenty-four hours of favorable ores, the cost per ton would be $3.54, which sum will be augmented by a certain proportion of the general expenses, as well as by whatever fluxes may be found necessary to add.

Where iron is cheap, the entire furnace is frequently inclosed in wrought or cast plates of that metal, the latter being usually seven-eighths inch thick and closely perforated with niche holes, cored out in the casting, to lessen the weight of the plates.

When thus supported, the exterior red brick casing may be omitted, as well as the massive corners, thus giving the main body of the furnace a coffin shape. This is the case with the reverberatories of the Orford Copper and Sulphur Company constructed by H. M. Howe.

ORE SMELTING FOR COARSE METAL.*

This operation requires the highest attainable heat in the shortest possible time. The furnace, therefore, should have a comparatively large fire-box and flue, and also a hearth of the largest practical dimensions, in order to contain a sufficient charge of the light and bulky ore.

Beginning with the old Swansea standard of 9 or 9½ feet width of hearth, the author has seen this gradually enlarged to 11 feet, without increasing the size of the fire-box (3½ by 4 feet) or the consumption of fuel, and raising the capacity of the furnace from 2 or 2½ tons per charge to 3¾ or 4¼, the time of fusion in both cases being identical; that is, from 4½ to 7 hours, according to quality of fuel and fusibility of charge.

Including delays, waste of time in charging, tapping, etc., 4 charges per day of 4 tons each may be considered good average work for calcined ores of reasonable fusibility, when the metal produced does not exceed 40 per cent. in copper. The production of a richer matte lengthens the period of fusion for several reasons.

Among these is the fact that rich matte does not possess the power belonging to low-grade metal, of rapidly floating up and detaching the semi-fused masses of ore from the furnace bottom, and thus shortening the process. Rich matte also presupposes a quite thorough calcination, so that there is not only a less proportion of easily fusible sulphides, but much of the sulphide of iron present has been changed into ferric, instead of ferrous, oxide, which is nearly infusible, and must be for the most part reduced to the lower oxide before it can combine with silica.

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* See paragraph entitled "Improvements in Reverberatory Smelting" for the latest results.

For these reasons, and more especially for fear of producing too rich a slag, the Swansea custom has forbidden the production of a matte from the first fusion of above 35 per cent. copper. The presence of arsenic and antimony, which require for their removal a long series of alternate oxidations and reductions, also has favored this practice; but with our purer and richer native ores, and with the high prices of fuel and labor, it has become the custom to produce a much higher grade of matte at the first fusion.

This also is the practice in Chili, where the first matte approaches 50 per cent.

Long experience in the practice of making a first rich matte has resulted in divesting it of many imaginary as well as real disadvantages. The following pages will show to what an extreme it has been pushed, where local conditions combine to render it advantageous.

The very high prices of labor and material at Butte, Montana, and the comparative cheapness of high-grade ores of copper, render that metal, while in the ore, about the cheapest thing there; so that it is much more profitable to sacrifice a small portion of copper than to attempt to save it all at the expense of fuel and labor.

Nor need the loss of copper in the slags from even the highest grade metal be as great as is often supposed.

The following results of work executed at the furnaces of the Parrot Copper Company, of Butte, Montana, illustrate, as has been said, the extreme practice of this kind. The assays were made by the company's chemist, Mr. D. P. Murphy.

Each assay represents a shipment of matte of from 15 to 16 tons, while the slag samples are average results of each charge, which could be extended to almost any desired limit.

Assay of matte. Per cent. copper.	Assay of slag. Per cent copper.	Assay of matte. Per cent. copper.	Assay of slag. Per cent. copper.
64·3	0·85	61·8	0·64
62·7	1·05	63·6	0·87
66·5	0·92	64·5	0·82
66·2	1·10	63·3	0·76
65·9	0·67	66·2	1·32

One of the great secrets of rapid fusion, always assuming that the flue and chimney are of the proper size, and that the firing is managed with skill and regularity, is the absolute exclusion of all currents of air from the interior of the furnace. The chilling effect produced by even the slightest crack or orifice in the brickwork or from deficient claying up of the doors, would be incredible to the inexperienced. The front and side doors, made of fire-brick, inclosed in an iron frame, are fitted closely to the sides of the furnace surrounding the door openings, and all cracks luted with plastic clay. As the charge is usually too siliceous already, and as any addition of this clay, which is sure to fall down upon, and become mixed with, the ore charge, at the side door, is a decided detriment, the smelter uses the smallest practicable quantity of the same. This shrinks on drying, permitting the ingress of air currents, which may have a most serious effect in retarding the process.

The author has remedied this by substituting for the worthless clay the raw slimes from the settling-pits, where concentration is employed. or fine screenings from the roast-heaps where the former material cannot be procured.

This may seem a small item, but the quantity of lute used is very great, as may be inferred from the fact that the substitution of the concentration slimes for clay in works containing six reverberatory furnaces effected an increased production of 1,000 pounds of copper monthly at no expense, but with the saving of some $15 in clay.

As the gangue of copper ore is usually quartzose, the reverberatory smelter is seldom troubled with too basic a charge.

Where this occurs, it is a decided evil; for although extremely fusible, it rapidly destroys the fettling and sand bottom, and, what is of far greater importance, produces so thin and fluid a slag that its removal from the metal by skimming is almost impossible. In such instances, the only remedy is to allow the liquid charge to cool until the slag has stiffened sufficiently to render its removal less difficult, or to throw a considerable quantity of ashes or coke-dust over the surface of the bath, which acts in the same way. It is sometimes necessary, after skimming the principal portion of the slag, to allow the remainder to chill until it becomes so thick that, on tapping, it will remain in the furnace, while the more fusible matte is run into the ordinary sand molds. This is a misnomer, however, as sand would be the worst possible material for the construction of molds, a sandy loam being the proper substance, and being dampened only sufficiently to render its manipulation feasible. It must be thoroughly dried before tapping, to avoid explosions, especially when coarse metal (below 45 per cent.), which is peculiarly liable to this accident, is being made. The richer grades of matte may be run into quite damp molds with impunity. But special care must be taken with blister copper, which is more explosive even than the lower grades of matte. If serious boiling of the liquid metal in the molds occurs, indicating a probable impending explosion, the spot should be at once covered with old boards, which should always be held in readiness. Any outbreak is controlled by dry sand, water being avoided in all cases. Fortunately, these demonstrations are less dangerous than they appear, and very few cases of explosion are so serious as to prevent the attendants from remaining in the building and protecting it from destruction by fire. A little pluck and plenty of dry sand will nearly always suffice to prevent any serious results.

After the removal of the pigs of matte from the molds by the furnace-helper, assisted by a second man, the sand should be raked over with an iron rake, and all coarse pieces returned to the next charge.

The pigs of slag from the slag-bed may be generally thrown over the dump, but the plate slag should be re-smelted entire, and every pig of slag, when cool, should be carefully examined for prills.

It is more advantageous to charge a furnace by the side door than by the means of a hopper above the roof, as the proper leveling and distribution of the heavy charges now used are almost impracticable by means of the rabble, while, when charged with a shovel, every pound of ore can be thrown just where it is needed. In order that no time be wasted, the helpers from other furnaces assist in charging, at least four men being required. The work is exceedingly hot and laborious, as the entire process should be completed in from ten to fifteen minutes, to avoid waste of time and fuel. Later experience has shown that charging can be done perfectly well from hoppers.

The tapping of the metal should occur as seldom as possible, as the influence of the molten matte upon the fresh charge is very favorable, and prevents that persistent adherence to the bottom that is one of the chief causes of delay.

In case a charge should adhere in this manner, it is usually better to skim it as soon as a few hundredweight of clean slag can be obtained. If the direct contact of the flame for half an hour or more still fails to raise the old charge entirely, the work should not be unreasonably delayed, but the fresh charge should be distributed in such a manner as to leave bare those portions that adhere most closely, and which will usually be loosened by this double period of firing.

Those portions of the hearth subject to the most excessive heat and wear, such as the bridge and side walls, should be thickly covered with ore, even to the extent, if necessary, of heaping three-fourths of the charge upon a comparatively limited area, if such practice be found conducive to the quickest fusion and greatest capacity.

In charging a mixture composed of various ingredients, the succession in which they are thrown upon the hearth is by no means a matter of indifference. With a mixture of calcined pyrites (or matte), raw quartzose ore, and rich slag (a very common charge), the calcined ore should be thrown upon the hearth, which it protects by its want of conductivity; the quartzose ore should come next; while the very fusible slag should surmount the whole. In this way, want of conductivity of the calcined ore is prevented from delaying the fusion, as it would if it covered any of the other substances, and is made positively useful in protecting the hearth.

The size to which ore should be crushed for reverberatory smelting depends upon its fusibility; very quartzose ore being benefited by passing a 4-mesh screen, while basic or sulphide ore may be of almost any reasonable size.

Very fine crushing should be avoided, on account of excessive formation of flue-dust as well as of its property of becoming so compact as to resist the highest temperature.

After fusing any calcined ferruginous material for several days, the hearth will be found covered with slimy masses of reduced iron, which, to a certain extent, may be beneficial as a protection to the bottom, but when beyond a certain limit, must be removed by persistent firing, assisted, when necessary, by a small charge of raw sulphurets, which will rapidly float up and dissolve the accretions. Several thousand pounds of metal are often obtained in this manner from an apparently empty furnace.

Every metallurgist should be capable of personally judging of the condition of his furnace bottom, as the shrewd smelter may gain great credit for speedy smelting by skimming his charges before they are really completed, while the honest furnace-man who waits until his hearth is clear before throwing the new charge may receive undeserved blame.

The amount of detail connected with the management of a reverberatory smelting-furnace is almost endless, and while it may be an easier task to manage a reverberatory than a blast-furnace, for an inexperienced man, it is an infinitely greater attainment to be a thoroughly skillful reverberatory furnace-smelter than to have equal skill in the management of the blast-furnace.

IMPROVEMENTS IN REVERBERATORY SMELTING.

During the three years that have elapsed since the publication of the first edition of this book, there have been considerable advances made in reverberatory practice. Furnaces have been enlarged, improved methods for the removal of the slag have been adopted, and decided advances have been made in other directions.

We are indebted to the Anaconda Works of Montana for some of these advances, but it is to Mr. Richard Pearce, of the Boston & Colorado Works at Argo, Colorado, that we owe the greatest gratitude in this direction.

While there are several minor points of construction, and various points of practice that are not yet made entirely public, yet I can in the main describe these improved reverberatories, and the means by which their capacity has been increased from some 16 tons to over 28 tons per day.

In the first place, the size of the hearth of the furnace has been increased from about 10x15 feet to 14x24 feet. The hearth has also been materially enlarged by shaping it differently, and not drawing it gradually to a point at the flue-end as in the older practice, but keeping it nearly its full width until close to the flue, and then contracting it rapidly.

The size of the fire-place has been increased by some 10 inches in length and 6 inches in width; and although the amount of fuel used is of course larger than in the old furnaces, yet it has by no means increased in proportion to the additional capacity gained.

As the smelting-charges now amount to some 6½ or 7 tons of ore, and the amount of slag is very great (especially in the Argo practice, where only about one ton of matte is produced to from 12 to 18 tons of ore), it was found necessary to adopt some rapid and economical method of disposing of this great quantity of molten material. This is accomplished at Argo, as shown in the figures, by running it directly out of the furnace building in cast-iron spouts.

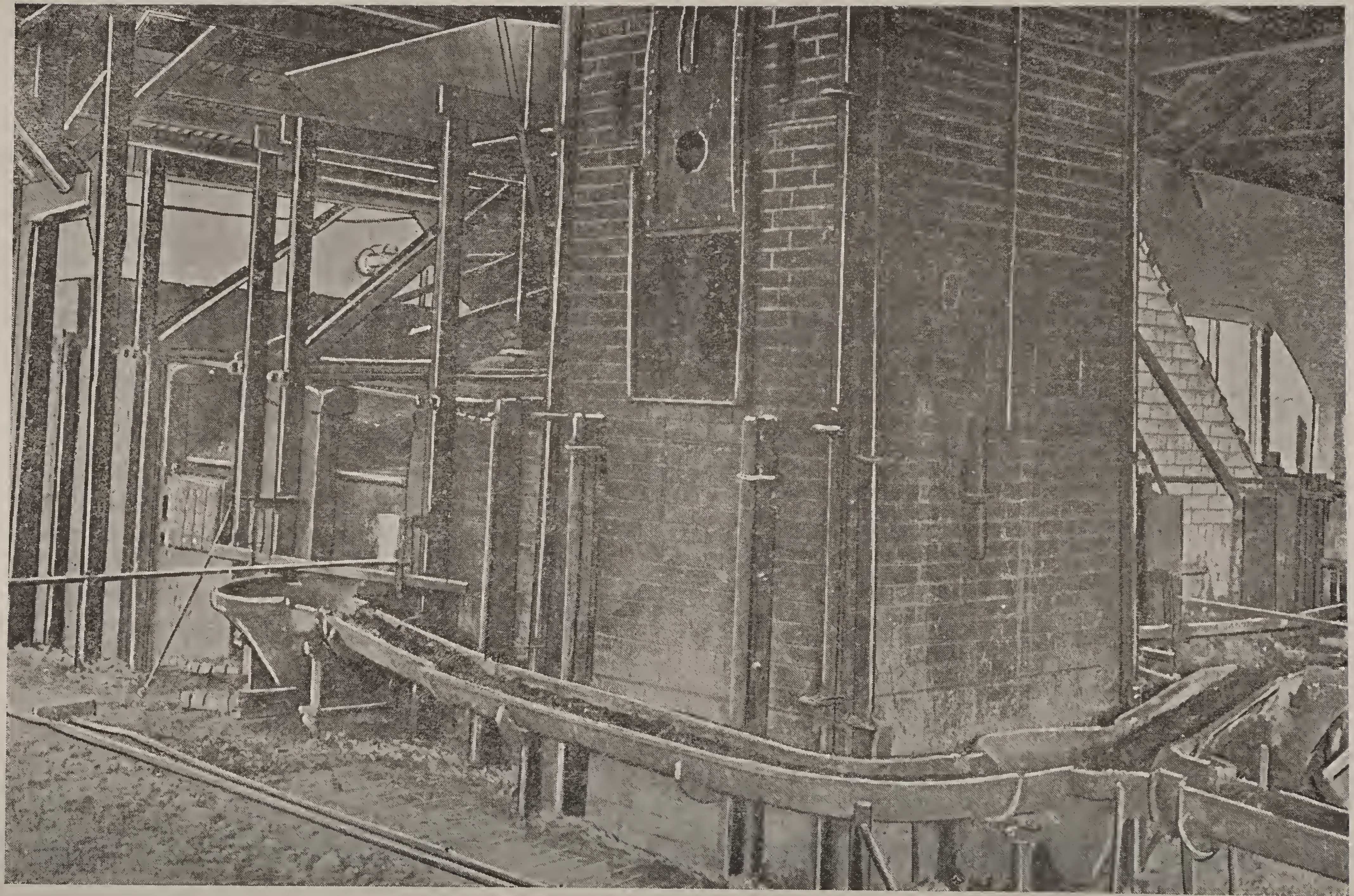
To lessen the time required for skimming, this operation is executed from both the front door and the side door, opposite the tap-hole. The slag from each of these doors is skimmed into a conical pot, which answers as a settling-pot for the grains of metal, while the iron gutters lead from each of these pots to the front corner of the furnace on the same side as the side skimming-door. Here they unite in a third conical safety-pot, while from this pot another length of gutter carries the slag outside the shed, the length of it that crosses the working-space in front of the furnace being hinged at one end and counterweighted, so that it can easily be hoisted out of the way when not in use.

To make this system perfect, this end-gutter should discharge direct into a large slag-car, or into a powerful stream of water that would granulate and remove the slag, the granules being subsequently removed into cars or to a high dump by means of an automatic bucket-elevator, but the conditions at Argo were not suited to such an arrangement, and the slag is simply run into sand-molds in the usual manner, and then loaded on railway cars, to be used as ballast for the line.

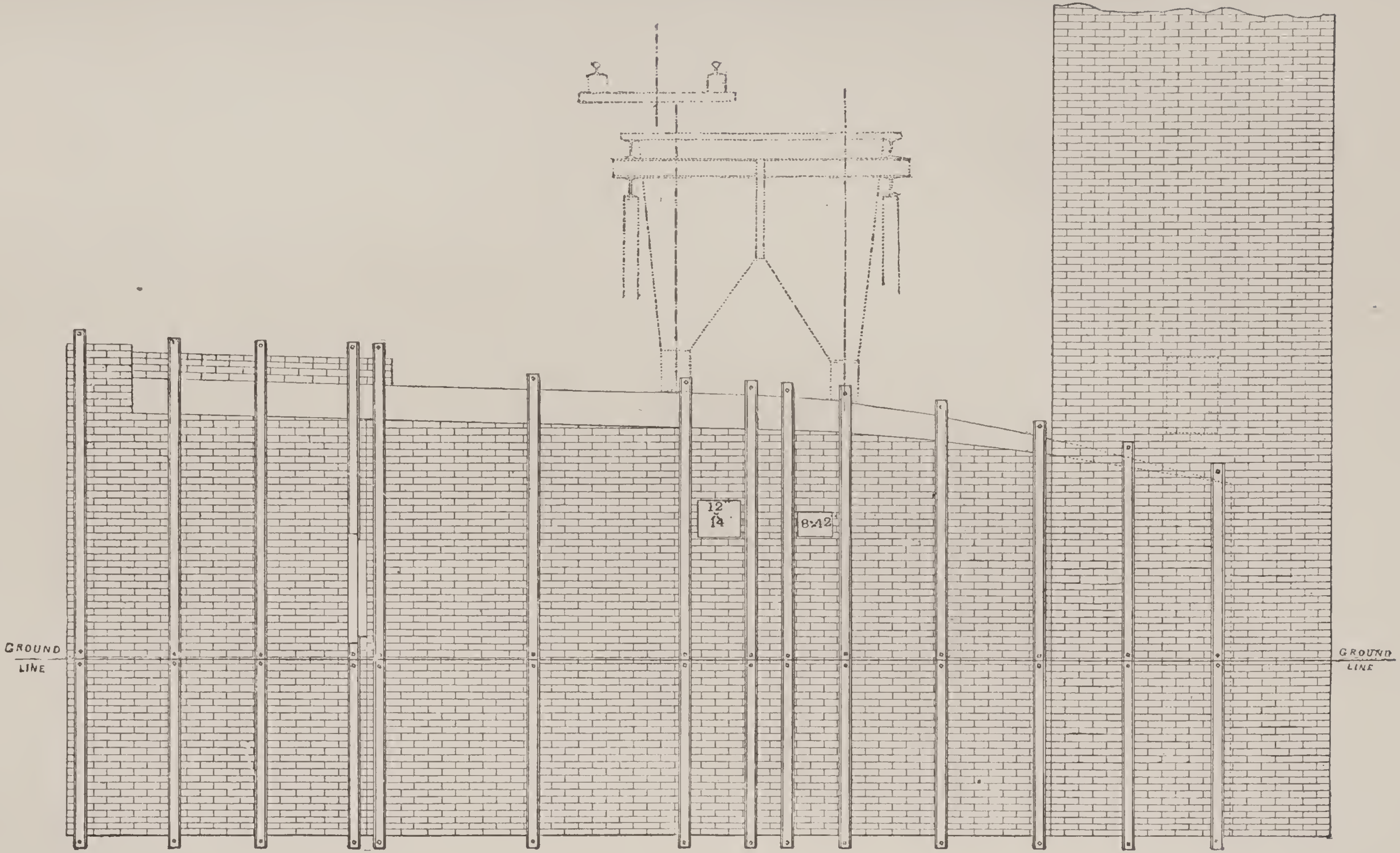
All these arrangements are so simple and obvious, now that they are once in operation, that they require little explanation.

The gutters being very heavy, thick castings cool the slag so rapidly that it does not weld to them or damage them, and they consequently require no clay-lining, and last a long time. The matte also is tapped into iron molds, by which all adherence of sand is prevented, and less slag is made in the next operation, besides the great advantage of always having a clean shed, unencumbered by a vast mass of heated and dusty sand.

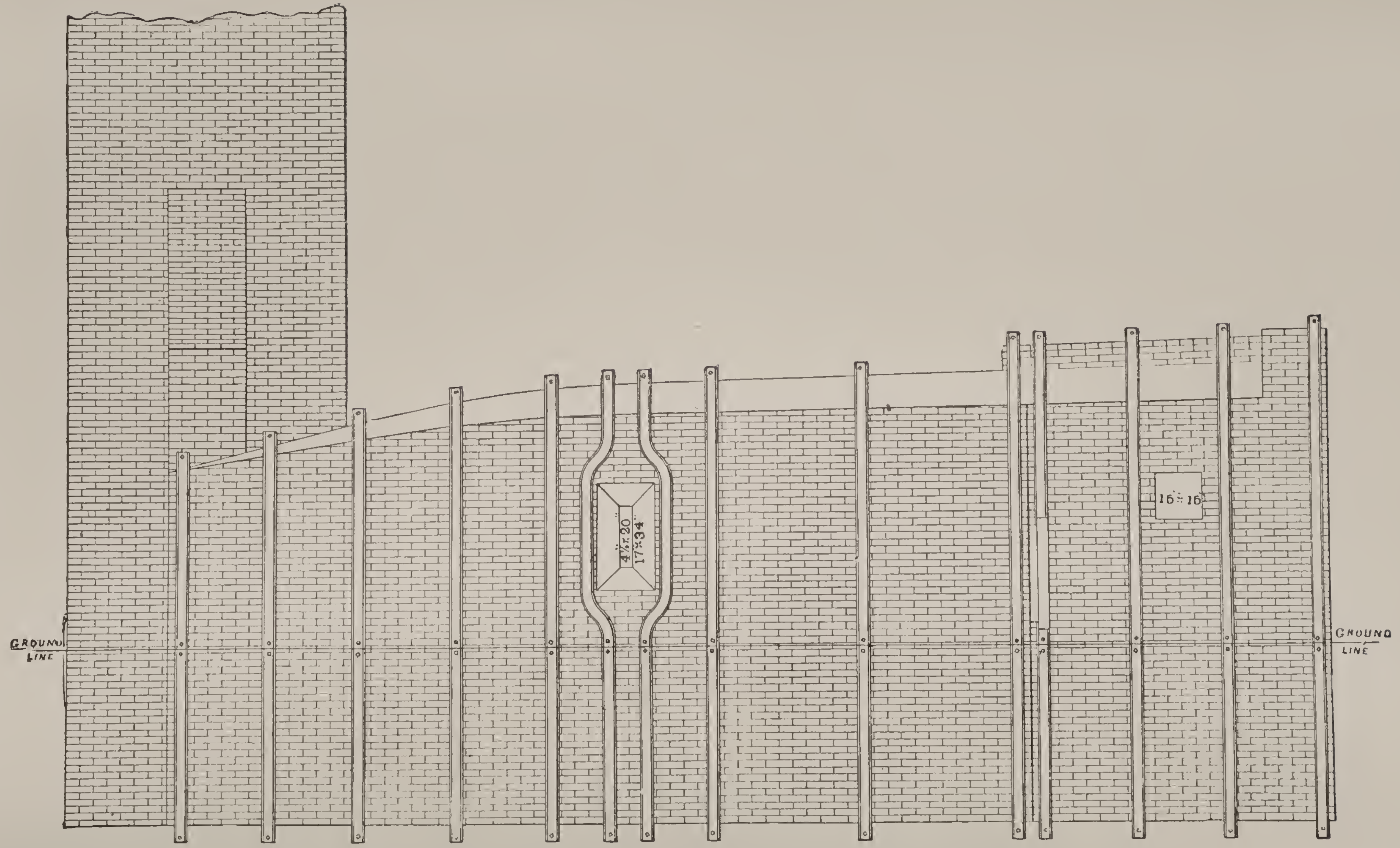
There is nothing new in preheating the air necessary for combustion before letting it pass under the grate-bars, but at Argo this preheating is more thoroughly and systematically executed than at any other works with which I am familiar. The cold air



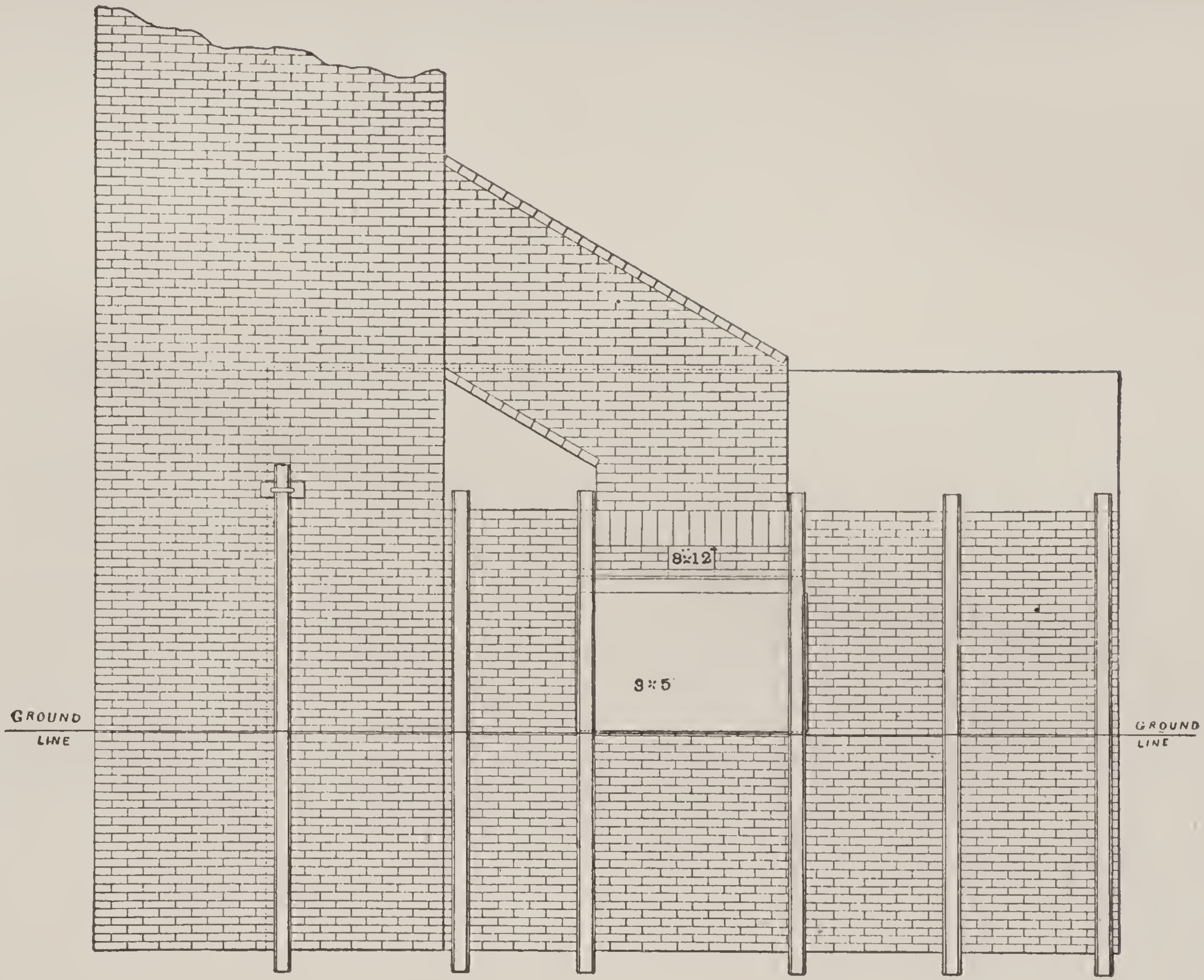
VIEW SHOWING POSITION OF FRONT AND SIDE SKIMMING DOORS, ALSO THE ARRANGEMENT OF SLAG SPOUTS.



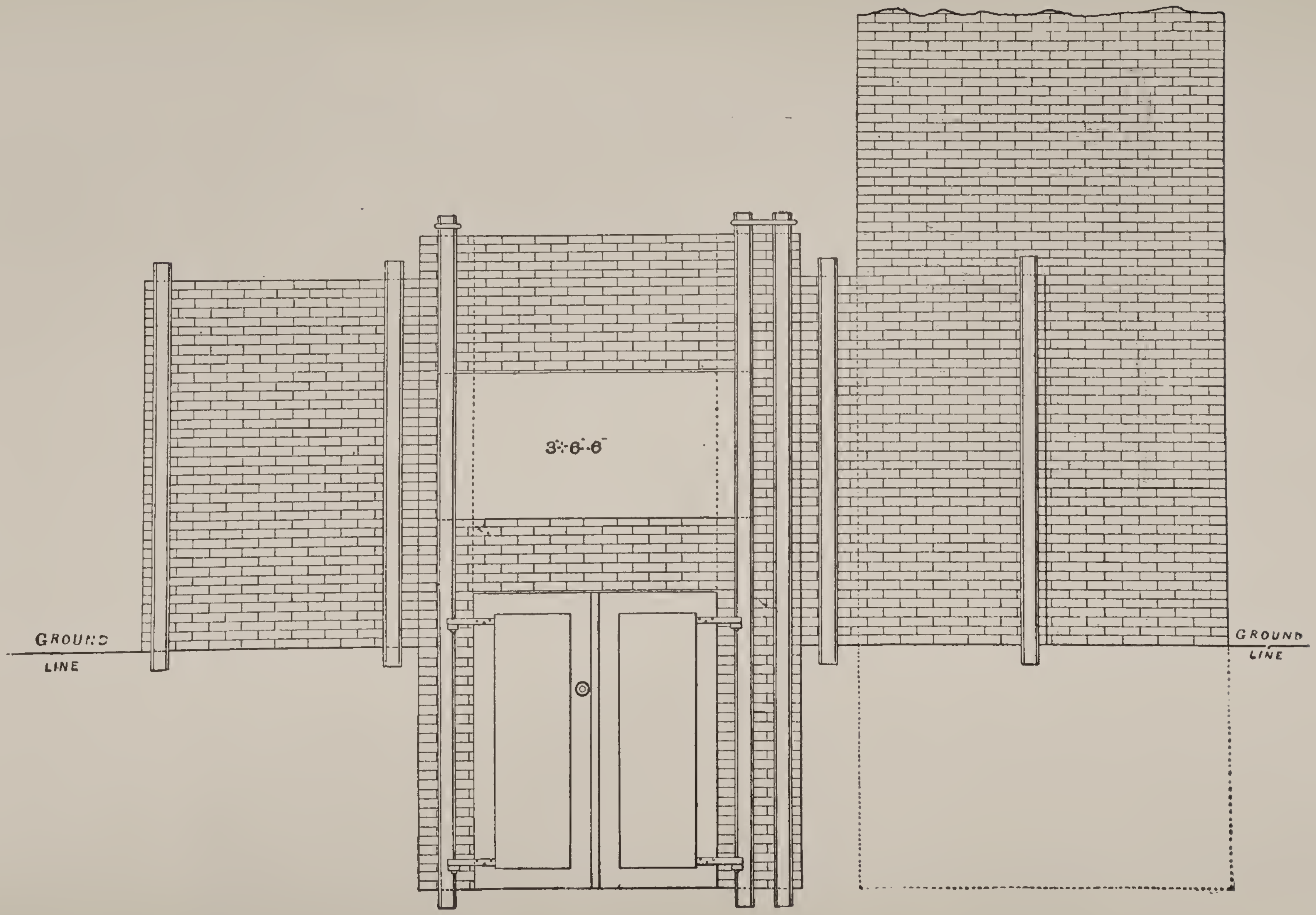
LARGE REVERBERATORY FURNACE. Side Elevation.



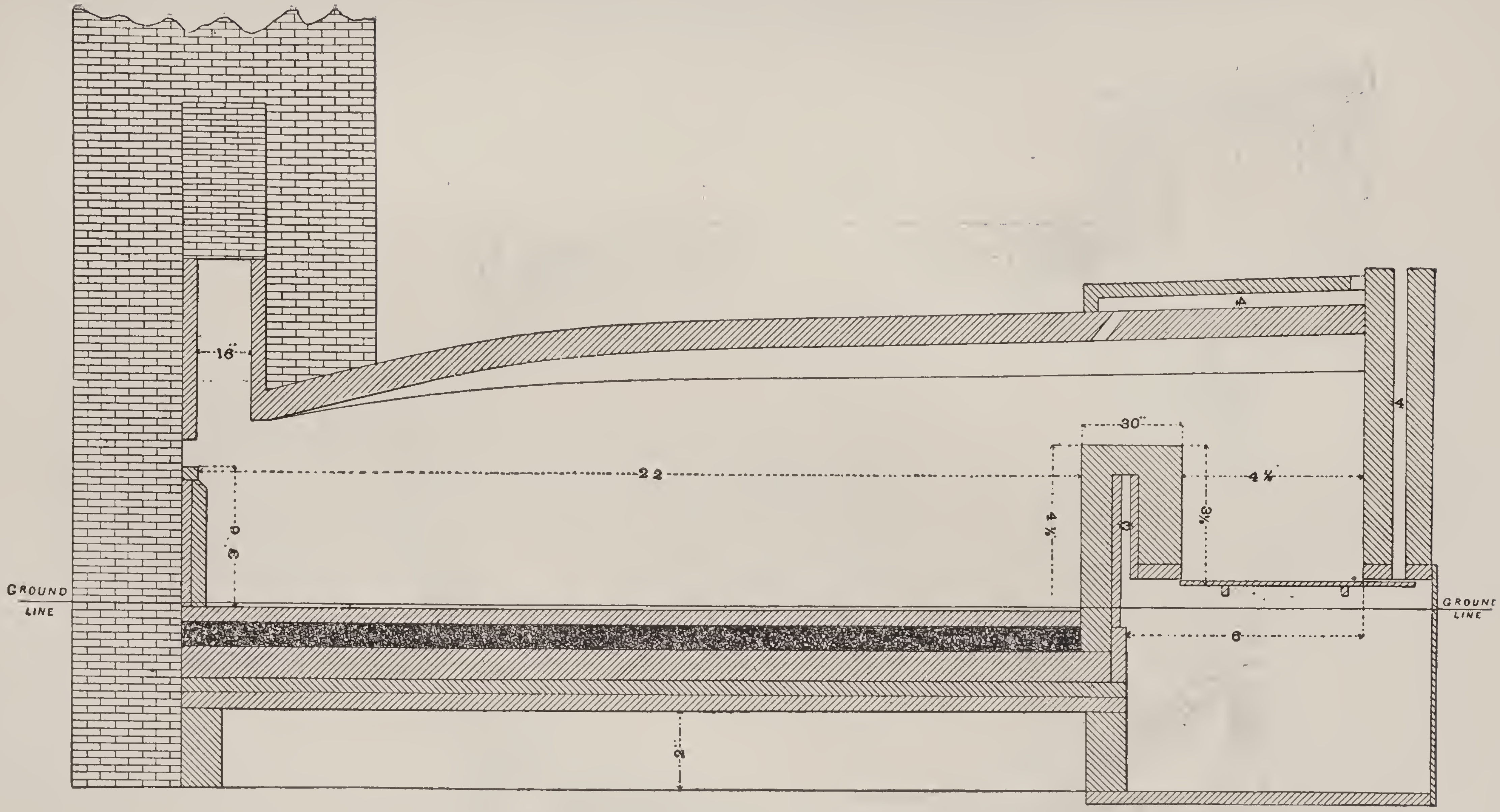
LARGE REVERBERATORY FURNACE. TAP-HOLE SIDE. Elevation.



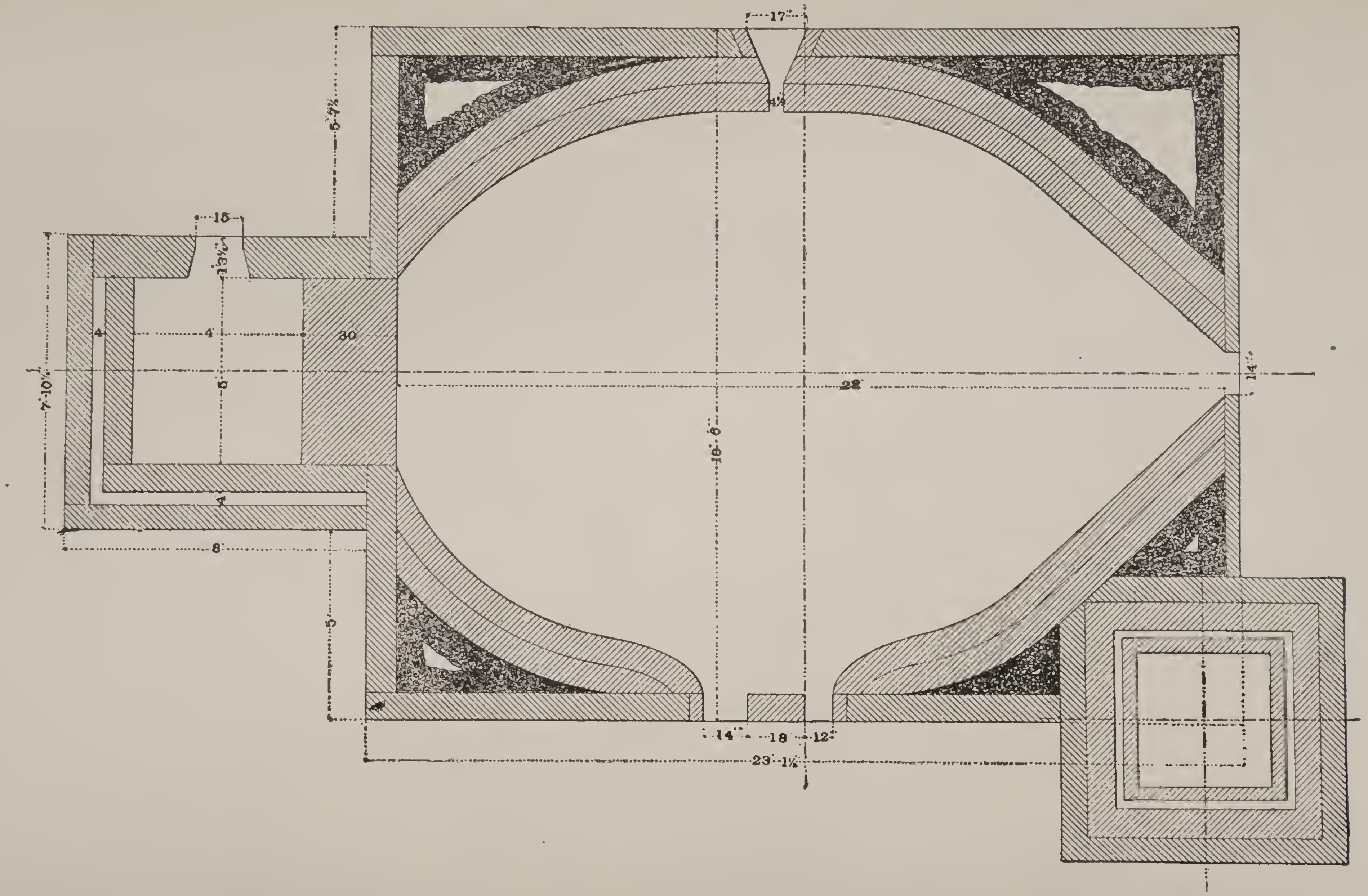
LARGE REVERBERATORY FURNACE. Front Elevation.



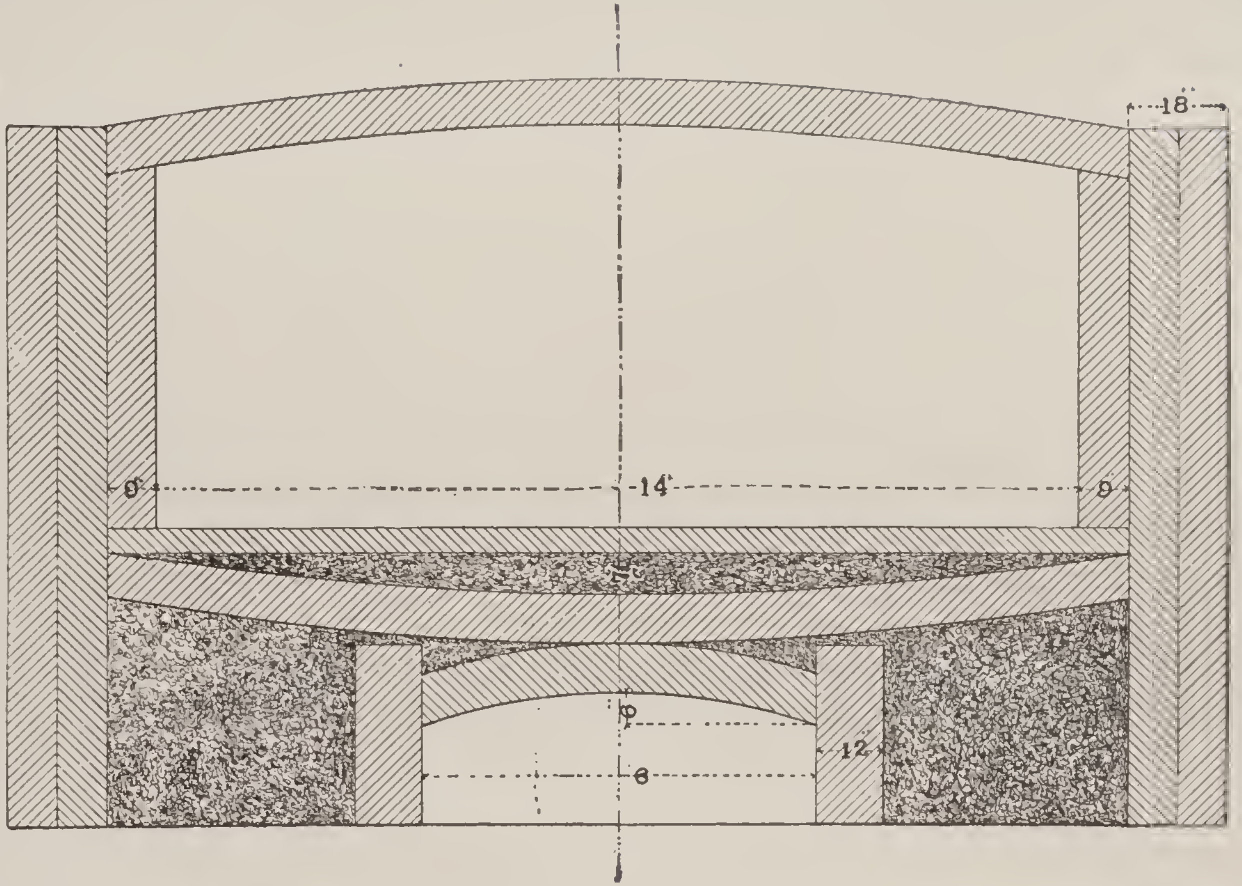
LARGE REVERBERATORY FURNACE. Rear Elevation.



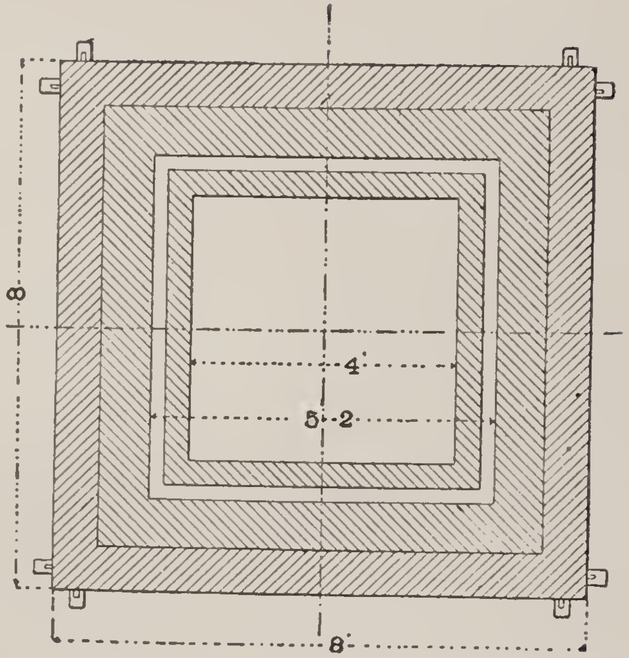
LARGE REVERBERATORY FURNACE. Longitudinal Section.



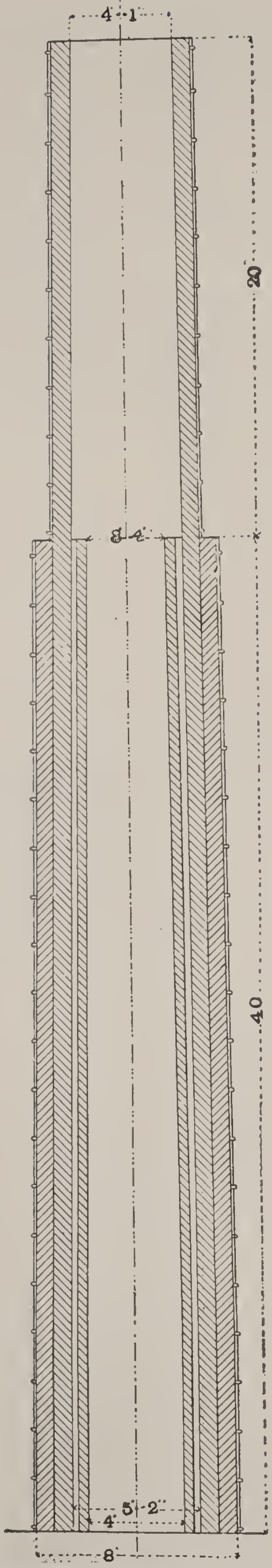
LARGE REVERBERATORY FURNACE. Sectional Plan.



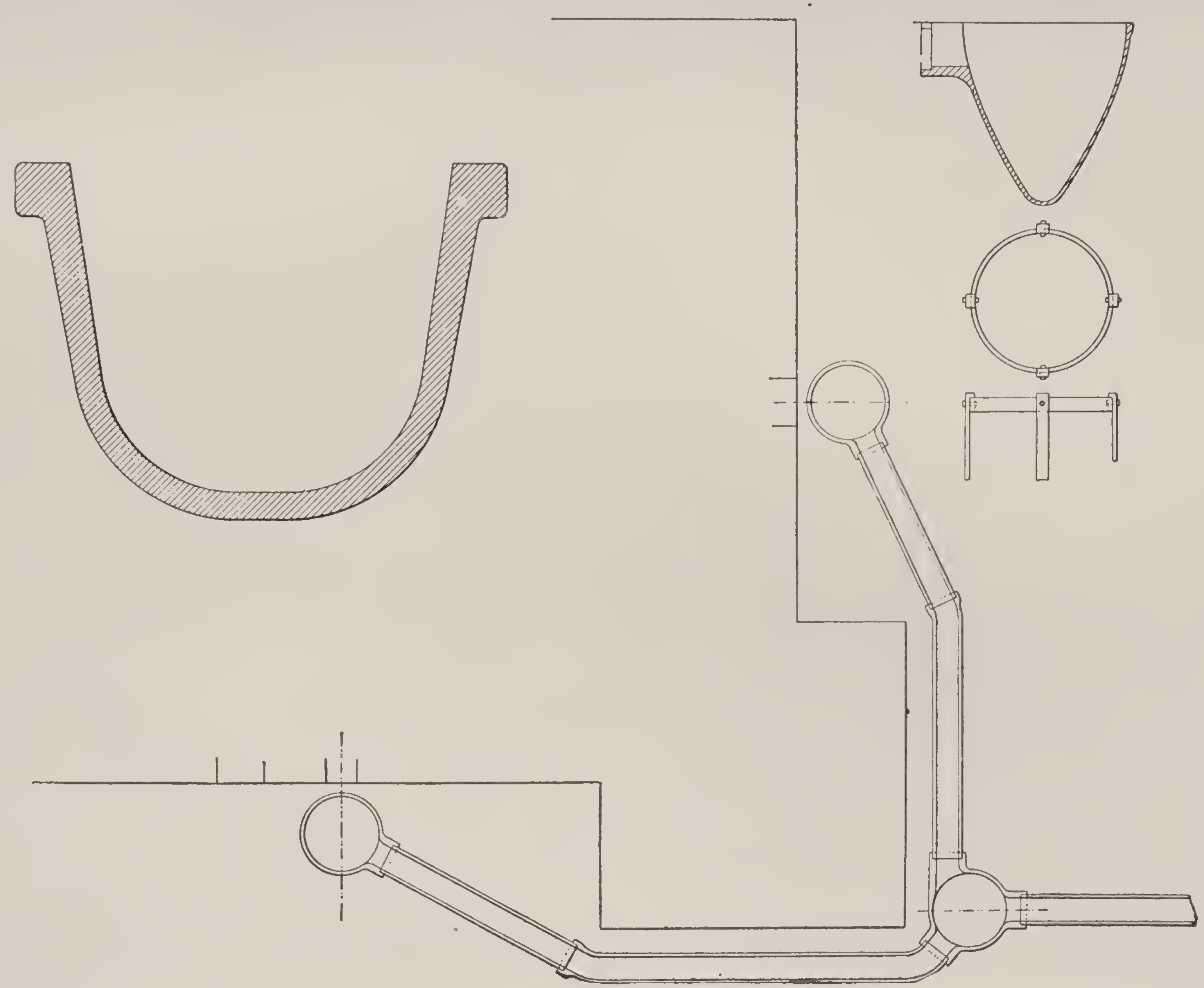
LARGE REVERBERATORY FURNACE. Cross Section.



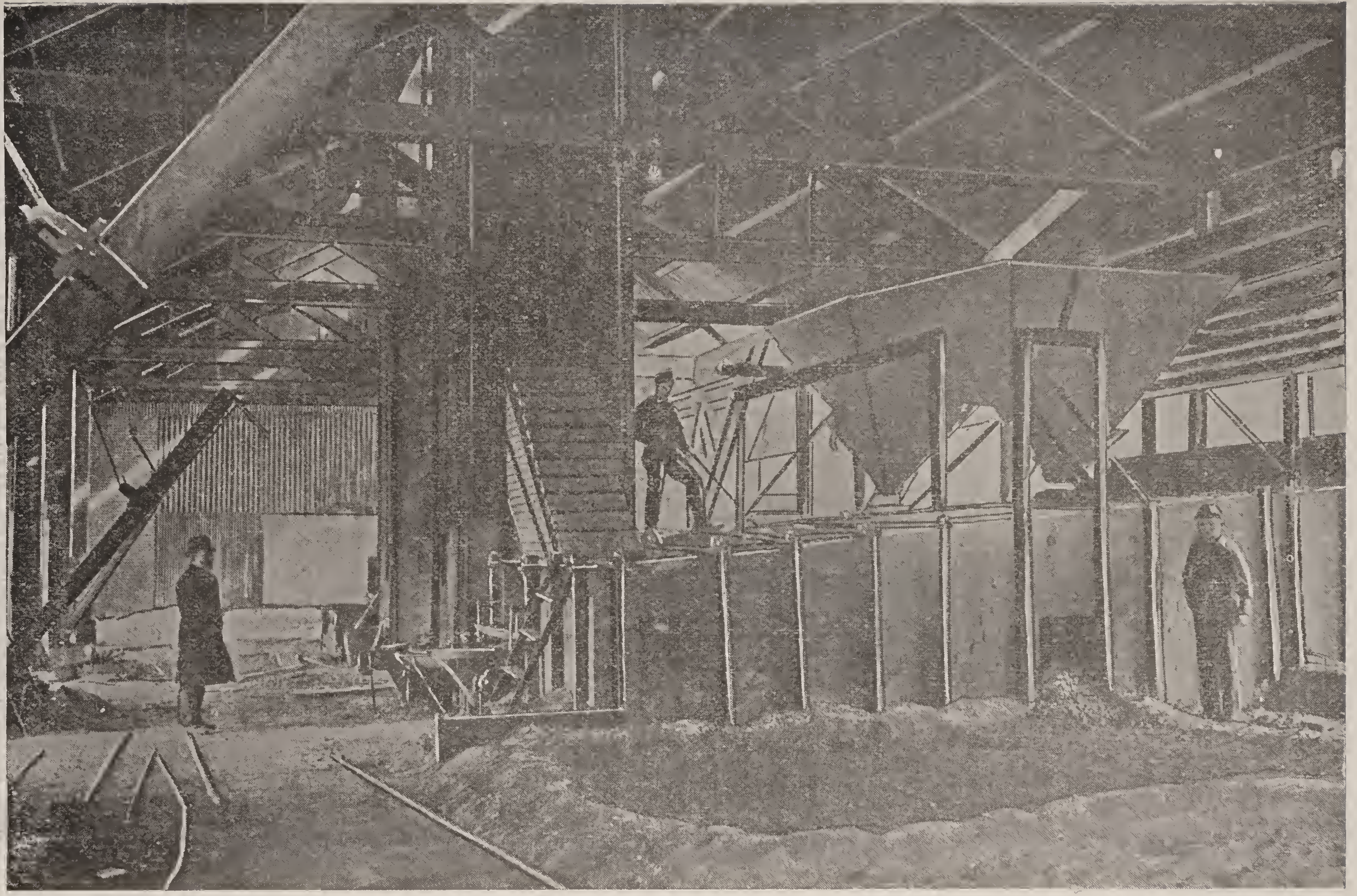
STACK. Cross Section.



STACK OF LARGE REVERBERATORY FURNACE. Longitudinal Section.



LARGE REVERBERATORY FURNACE. Skimming-Pots and Slag Gutters.



VIEW OF FURNACE FROM TAP-HOLE SIDE.

enters into channels formed in the brick-work near the front end of the furnace on each side, which openings are guarded by gratings. Thence it passes along deep side-walls, and close to the hearth-lining, and before it reaches the exit under the fire-box, it has become heated to a considerable degree. This system also assists in cooling the hearth-lining, and thus lengthening its life materially. Aside from this, heated air is allowed to enter through channels at either end of the bridge-wall, cooling the latter, and assisting greatly in the combustion of the gases, where it enters the furnace at the bridge. These openings are regulated by a valve, and to show that the benefit was not imaginary, my attention was directed to the difference in the flame shortly after throwing fire, and when these openings were alternately opened and shut. When closed, the flame was red and dusky, and a great volume of smoke escaped from the stack, while, on opening them, the flame at once became yellow and elongated, and the escape of smoke was almost entirely prevented. Others, including myself, have built furnaces with similar valve-openings to assist combustion, but owing to the ignorance and prejudice of the workmen, they have soon become blocked up and fallen into disuse. But Mr. Pearce has insisted on this point, until the smelters themselves have become convinced of their advantage, and now could not be persuaded to neglect them.

A decided saving in time and fuel may be also effected in reverberatories that are running principally on calcined ore, by charging the red-hot ore from the calciners directly into the smelter, as is done at Anaconda, Montana.

But this arrangement involves a plant laid out in a peculiar manner, and in already established works it is often impracticable.

But in building new works, where the ground and conditions are favorable, it should always be so arranged. The calciners, whether Brückner's cylinders or ordinary reverberatory-roasters, should be placed on a higher level than the smelting furnaces, so that the cars of hot ore from the former can easily be dumped into the iron hoppers of the latter.

Each calciner should be so arranged as to deliver its roasted ore into the hopper of each and every smelter, for, if using cylinders, the ordinary charge of 10 to 14 tons will have to be divided among three or four smelting furnaces; while, if hand-calciners are used, it will require the combined charges of several of them to furnish enough ore for a single reverberatory charge.

The use of red-hot, calcined ore by no means prevents the simultaneous addition of a greater or less proportion of cold ore from other sources, such as heap-roasted ore, raw, dry ores, or even cold, calcined ores. But unless the hot calcined ore forms at least 50 per cent. of the total charge, it is hardly worth while to undertake this method of working, as there will not be sufficient time and fuel saved to make up for the delays and extra labor that it entails.

This plan has been used so little and in so few places, that observations are mostly wanting as to the exact results obtained. From the statements of two gentlemen who are now using it to a greater or less extent, we may infer that where the charge consists mostly of red-hot, calcined concentrates, a saving in time of some 15 per cent., and in fuel of 20 per cent. may be anticipated. But these statements are only given as guides, and are subject to correction. In a few months I hope to know more about this point, as I am at present planning a smelting plant where the main portion of the reverberatory charges will consist of red-hot concentrates from Brückner's cylinders, supplemented by a small addition of first-class lump ore, roasted in stalls.

But though the utilization of the red-hot ore may not always be possible, there are certain improvements that are open to nearly every metallurgist, especially where he is called upon to plan new works, and is not hampered by old plants laid out on what is usually known in the profession as "The Patent English Wheelbarrow System."

Among these economies may be mentioned in particular:

The charging of the furnace by means of a hopper, instead of the slow and extremely laborious practice of having the ore shoveled in through the side door. This plan not only saves the wages of at least one man to the furnace per shift, but also effects a far more important economy in time and fuel, for every manager must have frequently chafed over the long interval of time that elapses between the beginning of skimming and the final luting-up of the doors on a fresh charge. It is much less labor to spread the ore over the furnace-hearth as it falls from the hopper than would be imagined, and especially where red-hot ore is charged, as it flows in a sheet over the entire hearth almost as though it were a liquid. The presence in the furnace of the molten matte resulting from the last one or more charges is also of the greatest benefit in shortening the next period of fusion; as it not only keeps the fresh charge from sticking to the bottom, but it acts as a solvent in separating the particles of fresh ore and greatly hastening their fusion.

It follows, therefore, that the matte should be tapped as seldom as possible, it being frequently advantageous to tap it only once in 24 hours; although this must depend upon the richness of the ore and the proportion of sulphur it still contains, as well as upon the depth of the hearth, and its capacity to hold a large quantity of matte.

In these large furnaces, taking 5 to 7 tons of ore at a charge, and especially if the ore is poor in copper, the amount of slag produced at each skimming is so enormous, that Mr. Pearce's plan of having two skimming-doors—one at the side and the other in its regular position—greatly lessens the delay.

The regular charging-door at the side of the furnace may be arranged for skimming, or a separate opening may be made close to it on the same side.

The lining of a reverberatory stack is frequently burned out at short intervals, especially when the combustion of the gases does not take place perfectly in the hearth.

Although the protection of this lining by an air-current is nothing novel, yet it is found profitable to go somewhat farther in this direction than the ordinary practice dictates.

The chimney proper should be built some 15 inches larger inside than it is destined to be eventually; the inner lining being of 4½ or 9 inches of fire-brick, solidly tied in at intervals with the red-brick casing, to make a strong wall. Inside of this, and separated from it by a 3-inch air-space, should come the false lining, consisting of one thickness, or 4½ inches, of fire-brick. This forms a perfectly independent shaft within the main chimney, and only connected with the latter by an occasional cross brick for steadiness.

By means of several ventilation holes through the casing of the stack below the flue, this air-space is placed in communication with the external air, and a powerful current of cold air is established throughout its entire height, which more than doubles the life of the false lining, and makes it exceedingly easy to renew it when worn out.

ESTIMATE OF MATERIAL AND LABOR ON NEW, LARGE REVERBERATORY FURNACES.

Although the latest type of reverberatory furnace does not differ from the old pattern, except in size, yet it may be useful to have at hand detailed estimates of the former, as well as full drawings, which accompany these estimates, and which may be relied upon as working drawings that may be used directly in construction.*

It will, of course, always be borne in mind, that different ores and different varieties of coal require differently proportioned furnaces; but under ordinary conditions, and with a fair quality of bituminous or lignite coal, the measurements here given will be found satisfactory.

The amount that may be smelted in one of these large furnaces, depends entirely upon the particular conditions of each case. With favorable sulphide ores, calcined so that not too much of the iron is changed into a sesquioxide, and so that it may be charged red-hot direct into the furnace, and with the charging and skimming arrangements made practically automatic, as shown in the accompanying drawings, it is quite possible to smelt from 30 to 36 tons per 24 hours in this furnace.

In a furnace of about the same size at the Argo Works, Mr. Pearce informed me that he had on an average put through 29½ tons of ore per 24 hours for an entire month.

MATERIAL USED IN CONSTRUCTION OF REVERBERATORY.

BRICK-WORK.

Fire-Brick:				Cubic feet.
One 4½″ foundation-arch	22′	× 7′	× 4½″	58
One 9″ foundation-arch, inverted	20′	× 15′	× 9″	225
One 4½″ flat hearth-bottom	20′	× 16′	× 4½″	120
One 9″ lining, both sides hearth	46′	× 3½′	× 9″	121
One front wall	10′	× 6′	× 9″	45
Two skewback walls of hearth	46′	× 5′	× 9″	173
One bridge-wall	6′	× 6′	× 2½′	90
One rear-wall, opposite skim-door	8′	× 7′	× 13½″	63
Two side-walls of fire-box	14′	× 9′	× 13½″	142
Flue, doors, clamps, etc.				123
Lining of stack (see plan of stack)				1,040
Total				2,200

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* The drawings of the large reverberatory furnace that accompany this estimate were just completed for The Eastern Development Company, Limited, of Boston, and Cape Breton, N. S. And I am indebted to this company for the privilege of using them in this connection.

2,200 cubic feet of fire-brick, at 18 brick per cubic foot, make, allowing for breakage and waste, 40,000 brick, at $40 per thousand		$1,600.00
Shaped Fire-Brick:	No.	
Bull-heads	250	
Side-skewbacks	250	
End-skewbacks	50	
Jamb-brick	60	
Wedge-brick	50	
Soaps	50	
Splits	100	
Total	810	32.00
Dinas brick for arch, 10,000 (used also in lining, etc.), at $60		600.00
Red brick for furnace and stack, 60,000, at $8		480.00
Fire-clay, raw and ground, 8 tons, at $8		64.00
Fire-clay, burnt and ground, 8 tons, at $8		64.00
Lime, 80 barrels, at $1		80.00
Cement, 80 barrels, at $1.50		120.00
Sand, 30 loads, at $1.50		45.00
Total masons' materials		$3,085.00

IRON WORK.

Cast-Iron:				Pounds.	
One bridge-plate, ribbed and holed	6′	× 3′	× 3½″	2,500	
One rear bridge-plate	6′	× 26″	× 2″	1,000	
Split plate for rear wall, ribbed	6½′	× 9″	× 1½″	600	
Plate to support rear of bridge	6½′	× 9″	× 1½″	600	
One front-plate, strongly ribbed	6′	× 2′	× 1½″	1,100	
One plate for end of fire-box	6½′	× 3′	× 1″	950	
Two skimming-blocks, each	26″	× 6″	× 4″	620	
Two door-frames				300	
Two skewback strips, in lengths	22″	× 4″	× 1″	2,200	
Three bearing and grating-bars	6′	× 2″	× 3″	432	
Gutters and pots				2,850	
Miscellaneous				1,200	
Total at 2½ cents per lb.				14,352	$358.80

Wrought-Iron:				
Tie-rods of 1¼-inch round-iron:	Feet.	Pounds.		
20 main furnace side-rods at 18½′	370			
8 cross fire-box rods at 8′	64			
2 longest rods at 31½′	63			
4 long main-hearth rods at 23½′	94			
Hooks, loops, etc.	232			
Total	823	3,410		
Carried forward		3,410	$358.80	$3,085.00

Brought forward	3,410	$358,80	$3,085.00
Iron for Stack:			
8 uprights, ¾″ × 1″, 60′ high, at 2½ lbs. per ft.	1,200		
100 cross-straps, 2″ × ¼″ × 8′ long, at 13½ lbs. each.	1,350		
Extra clamps for outside of stack	1,200		
Total at 2 cents per lb.	7,160	$143.20	

Buckstaves, old rails, say at 70 lbs. per yard:

[These need only reach just below bottom of fire-brick hearth, though shown deeper in drawing.]

Price, $25 per ton:		Ft.	Lbs.		
20 for main part of furnace	6½′ high,	130			
4 for sides of fire-box	8′ "	32			
2 for end of fire-box	12′ "	24			
4 for upper shoulder	8′ "	32			
6 for front end	5′ "	30			
Bearing-bars, etc.		50			
Total		298	7,000	$87.50	
Total of iron-work					$589.50

Miscellaneous:		
Lumber for patterns of arches, etc., 3,000 ft., at $18	$54.00	
Hoppers, complete	140.00	
Excavation and stone-work	300.00	
Grading and clearing up	70.00	
Total		$564.00
Labor:		
200 days masons' labor at $4	$800.00	
220 " helpers' " $2	440.00	
10 " carpenters' labor at $3	30.00	
10 " smith and helper at $5	50.00	
Total		$1,320.00
Incidentals and superintendence		$220.00

RÉSUMÉ OF TOTALS.

Masons' materials	$3,085.00	
Iron-work	589.50	
Miscellaneous	564.00	
Labor	1,320.00	
Incidentals and superintendence	220.00	
Grand total		$5,778.50

SMELTING FOR WHITE METAL.

As the production of the higher grades of matte, of which *white metal* (from 70 to 75 per cent.) may be regarded as the type, by means of the fusion of calcined coarse metal with quartzose ores, presents no sufficient differences from ore smelting to demand especial notice in this very brief treatment of the subject, the older process of concentrating metal without the intervening calcination need be alone considered under this head.

This process is termed "roasting" by the English smelter, and denotes the gradual fusion of the coarse metal in large pigs, on the hearth of a reverberatory furnace, with the abundant admission of air.

It is seldom practiced in the United States on account of its extreme slowness and consequent great consumption of fuel and labor, but it possesses the advantage of great simplicity of plant, dispensing as it does with the entire crushing and calcining paraphernalia.

Despite the simplicity of the process, much experience is required to obtain the best results, as the exact degree of temperature at the differing stages of richness of the product has much to do with the rapidity of the concentration.

Experience has taught that the rapidity of this concentration stands in exact proportion to the richness of the matte operated upon. The explanation of this is, that the sulphur, which is almost the sole foreign constituent of the richest matte, is very easy of oxidation, while the iron, which increases with the decrease of copper, oxidizes with much greater difficulty.

The writer has given much attention to this subject in connection with futile efforts to effect what M. Manhès has now accomplished with his Bessemerizing process. The following table gives, in per cent. of product, the result of his experiments, which extend over several years, many of them having been conducted for the Orford Copper and Sulphur Company, while the author was in its employ. They were made merely to determine the rapidity with which the grade of the matte increases by the ordinary method, and without any attempt at Bessemerizing.

Great care was taken in all instances to insure the correct sampling and assaying of the substances under consideration.

It will be understood that the matte was charged in the shape of large pigs; melted down during the time indicated (in most instances, about five hours), and retained in a molten condition (in both stages with the free admission of air) for varying periods, samples being taken from time to time—after thorough stirring—to determine the progress of the concentration.

TABLE OF MATTE CONCENTRATION BY OXIDIZING FUSION — PERCENTAGES OF COPPER IN FRACTIONS OMITTED.

Matte charged.	When fully melted. 5 hrs.	6 hours.	7 hours.	8 hours.	10 hours.	12 hours.	14 hours.	16 hours.	18 hours.	20 hours.	22 hours.	24 hours.	26 hours.	28 hours.	30 hours.	32 hours.	34 hours.	36 hours.	48 hours.
16	16	17	16	19	20	20	21	22	21	...	...	23	...	...	23	...	...	25	29
21	23	22	...	...	...	25	...	...	27	...	...	27	...	...	...	...	...	...	...
33	37	41	...	39	...	41	...	...	41	...	...	44	...	...	49	...	...	...	...
41	45	...	...	47	...	53	...	...	54	...	...	58	...	...	...	...	...	...	...
50	55	...	...	57	...	59	...	...	61	...	...	61	...	...	64	...	...	...	...
58	62	62	62	61	61	62	...	65	...	65	...	67	68	...	...	...	...	...	...
63	67	...	...	70	...	72	...	...	75	...	...	78	...	...	84	...	...	...	...
69	73	73	74	74	77	78	77	82	85	...	...	89	...	...	94	...	98	...	...
74	82	...	...	84	...	88	...	...	94	...	...	...	99	...	...	...	...	...	...
80	86	...	...	89	...	93	...	98	...	...	...	...	...	...	...	...	...	...	...
86	94	...	...	...	...	...	99	...	...	...	...	...	...	...	...	...	...	...	...
92	96	96	98	99	...	...	...	99	...	...	...	...	...	...	...	...	...	...	...
96	98	...	99	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...	...

THE MAKING OF BLISTER COPPER.

This very beautiful and economical operation is entirely of English origin, and though virtually belonging under the head of "Matte Concentration," presents so many important peculiarities as to demand separate notice.

The furnace used for this purpose, while an ordinary reverberatory as regards size and shape, should be very strongly ironed, to withstand the large charges used in our modern practice, while its bottom should be smelted in with peculiar care, and its upper layer should be thoroughly saturated, before used, with metal of the same grade as blister copper (from 96 to 98 per cent.), to prevent the certain annoyance from the rising of bits of the poorer matte just at the completion of the process, and the consequent adulteration of the whole charge of blister, which will require still further oxidation to remove the impurities. The lower bottom may be slightly saturated with lean matte to save expense.

The metal is charged in large pigs, the total weight depending principally upon its grade; for as a full bed of blister copper (from 6 to 10 tons) is usually desired as most economical, it is evident that a much greater weight of blue metal (62 per cent.) will be required than of white metal (75 per cent.); while pimple metal (83 per cent.) and regule (88 per cent.) will lose still less in the process.*

The technical names just enumerated apply to various grades of matte, each of which has its invariable characteristics, which distinguish it with certainty. The percentages given therewith are not absolute, but are subject to considerable variation, the writer giving such average figures as his own experience has determined for him.

As both economy and a due regard for the furnace bottom prevent the blister charges from covering too long periods of time, it is necessary to shorten the same by using either a less weight of matte, or insisting upon a higher grade at the outset.

The latter is the proper choice, as a small charge is almost certain to injure the furnace bottom by leaving a portion of it exposed to the direct heat of the flame.

The most advantageous lengths for the working off of a blister charge must depend largely upon local circumstances. From twenty-four to thirty-six hours will finish a full charge (eight tons) of rich pimple metal.

A similar weight of white metal may require fifty hours, which is quite long enough for the safety of the furnace, though a much greater length of time often elapses without harm.

As will be readily seen, it is impossible to work off a charge of blue metal within the prescribed limit of time, while even white metal extends the period most unpleasantly. It is therefore much better with metal of low quality to divide the operation into two stages—producing, for example, pimple metal or regule the first time, and bringing it up to blister copper by a second step. In this way, full charges can be used without endangering the furnace, and many advantages are gained.

It is not well, however, to alternate the operations in the manner just suggested; but rather to keep the furnace on one grade of metal until a large amount is collected, and then take up the blister process and maintain it until all the concentrated metal is disposed of. In this way, the evil of attempting to make blister copper on a bottom saturated with poorer matte is avoided; and the exact amount of concentrated metal required for a full charge of blister can always be had.

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* Since this was written, blister charges have been fully doubled in weight, the furnace hearth being made much larger than formerly thought practicable.

The matter may be pushed a step farther, by using a separate furnace for each operation, and positively interdicting the use of the blister furnace for any other purpose.

The operation of making blister copper is frequently executed by constantly maintaining the charge in a molten condition after it is once melted, and never allowing it to chill or "set," as it is technically termed.

By pursuing the latter plan, however, the danger to the hearth in long campaigns is greatly lessened, as it thus has a slight opportunity to cool, while the process is certainly advanced in a remarkable degree by the alternate fusions and chillings.

The belief that the operation of tapping causes a great gain in the grade of the matte, expressed among the Welsh smelters by the vulgar saying that " two tappings is worth one blowing," is contradicted by the following experiment executed by the writer for the purpose of determining the truth or falsity of the common belief:

	Just before tapping. Per cent.		After tapping. Per cent.
No. 1 Copper	53·6	No. 1 Copper	53·2
No. 2 Copper	66·8	No. 2 Copper	68·0
No. 3 Copper	72·7	No. 3 Copper	70·2
No. 4 Copper	75·4	No. 4 Copper	75·2
No. 5 Copper	79·3	No. 5 Copper	79·1
No. 6 Copper	85·4	No. 6 Copper	86·2
No. 7 Copper	94·0	No. 7 Copper	93·2
No. 8 Copper	98·7	No. 8 Copper	98·5
Average	625·9	Average	623·6

A few test assays that showed remarkable variation one way or the other were discarded, and, as seen, the others actually show, on an average, lower, rather than higher, grade of the metal after tapping, which is doubtless merely an accidental circumstance.

The exact grade at which the blister copper should be tapped is a matter of much importance, as either too high or too low a copper presents physical qualities injurious to the requirements of the process.

Blister copper of just the right grade is highly "red-short"; that is, can be easily broken when red-hot. It is this quality that enables the furnace-man to break and separate his pigs of blister copper, which operation is rendered greatly more difficult by a very slight variation in percentage in either direction. The proper condition of the charge is easily made manifest to the experienced by ocular inspection, and the writer has endeavored to fix the limits that bound the grade of copper possessing this important quality.

Foreign impurities exert so much influence in this direction as to render impossible any exact establishment of such boundaries; but numerous tests have fixed the most favorable grade between 96¼ and 99 per cent.

An important precaution in the care of a blister furnace is the proper draining of the hearth and stopping of the tap-hole. Carelessness in this respect will permit a slight and unsuspected leakage during the entire period of blister-making, culminating in a mass of metallic copper filling the entire tap-hole, and, as has occurred to the writer, requiring the combined efforts of the furnace *personnel* and blacksmith employés for twelve hours to remove it.

The charge, being high blister, and just ready to tap, exerts a ruinous effect upon the furnace bottom during any such delay, and should be ladled out at once under similar circumstances.

The slag from the blister process, being very rich in oxide of copper, should be returned to some process where the product is of high grade, and not, as is often the case in this country, sent back to the ore smelting, where its copper contents are thrown back again to the condition of a base sulphide.

The following are examples of the composition of the slag and copper produced by this roasting-smelting for blister copper, taken from Mr. Howe's paper already referred to:

	Welsh "roaster" slag.[*]	"Roaster" slags from Kaafiord.[†]		Welsh blistered copper.	Blistered copper. from Kaafiord.[†]
Silica	47·5	36·0	Copper	....	99·2–99·4
Protoxide of iron	28·0	7·0	Iron	·7–0·8	0·1– 0·2
Alumina	3·0	6·0	Nickel and cobalt	0·3–0·9	0·2– 0·3
Cuprus oxide, Cu_2O	16·9	43·2	Zinc	....	0·0– 0·02
Lime	....	2·7	Tin	0·0–0·7	....
Magnesia	....	0·8	Arsenic	0·4–1·8	....
Nickel and cobalt oxides	0·9	4·9	Sulphur	0·1–6·9	0·1– 0·12
Oxide of tin	0·3	0·6			
Oxide of zinc	2·0	3·2			

[*] Le Play. Op. cit., p. 218.

[†] Kerl, Grundriss der Metallhüttenkunde i., p. 215.

THE USE OF BASIC-LINED FURNACES.

Various attempts have been made at different times to construct the hearth and lining of blister-furnaces of some material that would have a purifying effect upon the copper treated therein, by absorbing deleterious constituents, and thus hastening the purifying process.

But none of these efforts have been sufficiently successful to gain any firm hold upon the profession, and my own experience in this direction has plainly shown me their futility.

Within the past two years, however, Percy C. Gilchrist, A. R. S. M., has made an elaborate series of experiments of a somewhat novel character, and that according to his own statements, as well as in the belief of many metallurgists who are qualified to judge of the matter, have led to a positive improvement and economy in that branch of copper-metallurgy known as the "Blister Process," as well, though in a considerably less degree, as in the actual refining of the blister copper.

Mr. Gilchrist published a full account of his discoveries in *The Journal of the Society of Chemical Industry*, published in England, in January, 1891, and while I intend this work to be mainly a record of my own experience, I feel it necessary to briefly point out the very remarkable results obtained by Mr. Gilchrist, and which I hope before long to be able to speak of from personal trial.

The most common serious impurity in copper is arsenic. This is especially the case in England, where our pure ores are the exception rather than the rule, and where the copper smelters are expected to make good copper from material carrying one, two, and even five per cent. of arsenic.

At the works where Mr. Gilchrist made his experiments, a large amount of highly arsenical copper-precipitate is treated, being added in varying amount to the slag and matte charged in the second smelting operation.

The mixture is so regulated that there is not sulphur enough present to combine with all the copper. Therefore, the product of a quick fusion of the above mixture consists of a considerable quantity of rich matte, comparatively free from arsenic, and a much smaller quantity of very impure metallic copper, known as "Metallic Bottoms."

Arsenic has a much greater affinity for metallic copper than it has for matte; hence by this simple operation, the main portion of the arsenic is concentrated into a small quantity of very impure "bottoms," while the great bulk of the copper is obtained in the shape of a matte of reasonable purity.

If gold were present, it would also concentrate very perfectly in the "bottoms," as is well known to all copper smelters, who use this method extensively in treating auriferous mattes.

An average analysis of these "bottoms" gives:

	Per cent.
Copper	83 to 87
Arsenic	5 " 7
Sulphur	1 " 3
Iron	0·5
Lead	3 " 5
Silica	0·5

It is in the conversion of these very impure "metallic bottoms" into blister copper containing under one per cent. of arsenic that Mr. Gilchrist's treatment seems to most plainly show its value.

Having been much annoyed, as are all copper refiners, by the enormous amount of rich slag resulting from the long, slow melting and refining of these bottoms into blister copper; and knowing this slag to be in the main a silicate of copper, the silica of which must come from the hearth and lining of the furnace, it occurred to him that if he should use a basic hearth and lining, the amount of slag should be greatly decreased, owing to the absence of any silica to combine with the oxide of copper.

Obtaining permission from the Company to make the experiment, he proceeded at once to put in a hearth of the same basic material as is used in steel furnaces; mixing the ground material with tar in the usual way, and burning it onto the bottom of the furnace in layers; each layer being well beaten down and fired for some hours, before applying a fresh one. This is a somewhat tedious operation, requiring some 4 or 5 days to properly burn in a basic bottom.

The taphole of the furnace is closed by throwing a little basic material against its internal opening, and the basic hearth absorbs in time just about the same quantity of copper as the ordinary quartz hearth.

Like the latter, it must be carefully "seasoned" by smelting a few light, quick charges of rich copper precipitate, or good blister copper upon it, till it has absorbed sufficient copper to make it solid and hard.

The repairs to the fettling of this furnace are very slight, as the tendency is rather for the fettling to grow than to cut away.

The practical merits of this new departure are emphatically shown by the fact that in a few months after the first basic furnace, the Company had nine of its blister furnaces lined in this manner, and intended to alter all the others as fast as they wore out.

The following table from Mr. Gilchrist's paper shows in a condensed form the extraordinary and favorable difference produced by the use of a basic lining. I will remind my readers, that in English metallurgical nomenclature, blister furnaces are called "roasters," and the operation of blister making is known as "roasting".

COMPARATIVE STATEMENT OF METALLIC BOTTOMS, ROASTED, RESPECTIVELY, IN BASIC- AND SAND-LINED FURNACES.

	Basic Furnaces.	Sand Furnaces.
Metallic bottoms charged during 12 weeks:	Tons of 2,000 lbs.	
Average analysis, 84·52% copper, 5·91% arsenic	448.56	448·56
59 charges made, averaging about	7·574	7·574
Blister copper produced from same	362·124	214.76
Average produce per charge, averaging 1·11% As	6·132	3·64
Slag made from above charges	115·178	248·36
Average weight of slag per charge	1·946	4·2
Average per cent. of copper in slag	25%	55%
Time occupied per charge, including fettling, etc.	29¼ hrs.	38 hrs.

A careful study of the above table will be found very instructive. In the first place, assuming the "metallic bottoms" that were charged into the furnaces to average 84·52 per cent. copper, and the blister produced to average 98·5 per cent. copper; it will be seen that from the furnaces with basic lining, the remarkable yield of 94 per cent. of the copper charged was obtained in the shape of blister copper; while from the ordinary furnaces, only 56 per cent. was obtained, a gain of 38 per cent. in favor of the basic furnaces.

This simply means, that by the use of the basic lining, the amount of slag to be re-treated from the blister process is reduced more than 50 per cent., there being only 115 tons of slag instead of 248, as produced in the acid furnaces. And when we come to estimate the amount of copper tied up in this slag, the advantage of the basic lining becomes still more apparent; for not only does it produce less than one-half the total quantity of slag, but this slag carries less than half the percentage of copper that the slag does that is produced in ordinary furnaces; being only 25 instead of 55 per cent. The actual amount of copper in the basic furnace slag is only 28¾ tons instead of over 136 tons as in the acid-lined furnace. Thus, only about one-fifth as much copper goes into the slag where the basic lining is used; which means that four-fifths of the copper that goes into the slag during the common blister operation is hereby saved, and goes at once to market without any further expense than has to be put on all the blister copper.

But this by no means represents all the advantage that arises from the employment of a basic hearth. It is a well-known fact to all copper-refiners that the production and treatment of these arsenical "metallic bottoms," and the long period that this immense mass of molten copper has to be kept in the furnace to permit the arsenic to become gradually volatilized, have a most destructive effect upon the hearth of the furnace, which usually requires quite extensive repairs after every few charges, and frequent, radical renewals.

This heavy expense has stopped the production of "metallic bottoms" at most of the British smelters, the managers finding it more profitable to charge their copper-precipitate into the smelting furnaces, together with sulphide ores, and thus "throw it back" into a matte.

But by the use of the basic hearth, this expensive repair bill is almost entirely suppressed, and the full advantages arising from the formation of "metallic bottoms" are obtained without the heavy costs commonly connected therewith.

TREATMENT OF WHITE OR PIMPLE METAL ON THE BASIC HEARTH.

Although such striking advantages are not obtained in smelting matte in a basic furnace as in the treatment of "metallic bottoms," yet a very considerable gain in output results therefrom.

Mr. Gilchrist gives the result of a long series of comparative trials by which it was ascertained that there was a gain of from 20 to 30 per cent. in the amount of copper produced in favor of the basic furnace. This means, of course, a very large saving in the subsequent treatment of the blister slag.

Arsenical copper frequently arrives at blister pitch, but still contains too much arsenic to tap, and has to remain molten under an oxidizing flame until the arsenic is reduced to the desired limits, usually under one per cent.

It is found that the use of the basic furnace favors the removal of the arsenic, and greatly shortens the period of oxidation, the relative time, in hours, being found on an average to be about 6 for the basic, and 10 for the ordinary, furnace, or a saving of some 40 per cent. of time in favor of the new method.

The use of the basic hearth in refining copper was also carefully investigated by Mr. Gilchrist, and his results and deductions will be found at the close of the chapter on copper refining.

COPPER REFINING.

The only method of copper refining practiced in the United States, or, in fact, in the civilized world, is the ordinary Swansea process.*

The purity of our local ores and the simplicity of our trade requirements have prevented the development of those marked variations that characterize the English process, where "best selected," "tough cake," "founder's metal," and many other distinct varieties are demanded and produced.

The great excellency of the Lake Superior copper, derived from pure metallic ores, has established a very high standard in our markets, and owing to the abundant supply of the same, manufacturers have not, until recently, found it necessary to study the behavior or familiarize themselves with the capabilities of other brands of copper for certain uses, but, feeling sure that if they bought the best they would be safe, have employed this unnecessarily superior metal for the manufacture of brass castings, and many other purposes where a poorer quality would have served equally well.

The most impure domestic coppers are often sufficiently argentiferous to repay a separate process by which the quality of the baser metal is improved, while the nobler is saved—of late largely by electrolytic means.

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* A few unimportant exceptions to this statement may still exist.

The refining-furnace presents no peculiarities to distinguish it from the ordinary English reverberatory, except that it should be more strongly constructed; being provided with a massive front plate—below the skimming-door—as well as strong horizontal, lateral braces to strengthen the hearth, which, in addition to the enormous expansive force caused by the high temperature, must also sustain the weight of from 10 to 15 tons of molten metal.

The ash-pit is very advantageously provided with iron doors, which may be closed during the ladling, to exclude all currents of air, while the flue is brought as nearly as possible over the skimming-door, in order that the air-current that enters therefrom may ascend at once without affecting the metallic bath.

The two bottoms are smelted in with unusual care, and the upper one thoroughly saturated with repeated small charges of metallic copper. This should be spread over the entire surface in the shape of granules, and should be rapidly fused until it is entirely liquid. At first, nearly all will be absorbed, but eventually, a larger and larger proportion will be regained, and thoroughly dipped from the ladling-hole at the close of each operation. A hearth of the ordinary size, 9½ feet by 14 feet, will absorb from 6,000 to 18,000 pounds of copper during the "soaking" process, according to the quality of the sand and the temperature attained during the "smelting in" of the bottom.

On no account should metal of poor quality be used for purposes of saturation. This is a fatal economy, as the grade of all copper refined in the furnace for many months may be affected thereby.

Even after the bottom is well saturated and has attained a considerable degree of firmness, the careful refiner will avoid any possible injury thereto from heavy masses of metal. It is not uncommon to charge pigs of blister copper weighing 1,000 pounds or more, the sharp corners and edges of which are very likely to cause indentations and unevenness in the toughest bottom, which may serve as the starting-point for serious after-effects. All such injuries may be avoided by laying down a rough floor of old planks or similar material.

The fuel best suited to refining is a not too caking bituminous coal with long flame. Sulphur-bearing coals should be avoided, as tending to alter the "pitch" of the metal at critical moments. Where such coal is expensive, a cheaper variety may be used for the earlier stages of the process.

No better fuel exists for refining than wood, as its freedom from sulphur and other impurities, and the long, pure, non-reducing flame that it yields, peculiarly fit it for the purpose. It was used entirely at the Ore Knob Refining-Works with great satisfaction, and would be more frequently employed were it cheaper at the great copper centers.

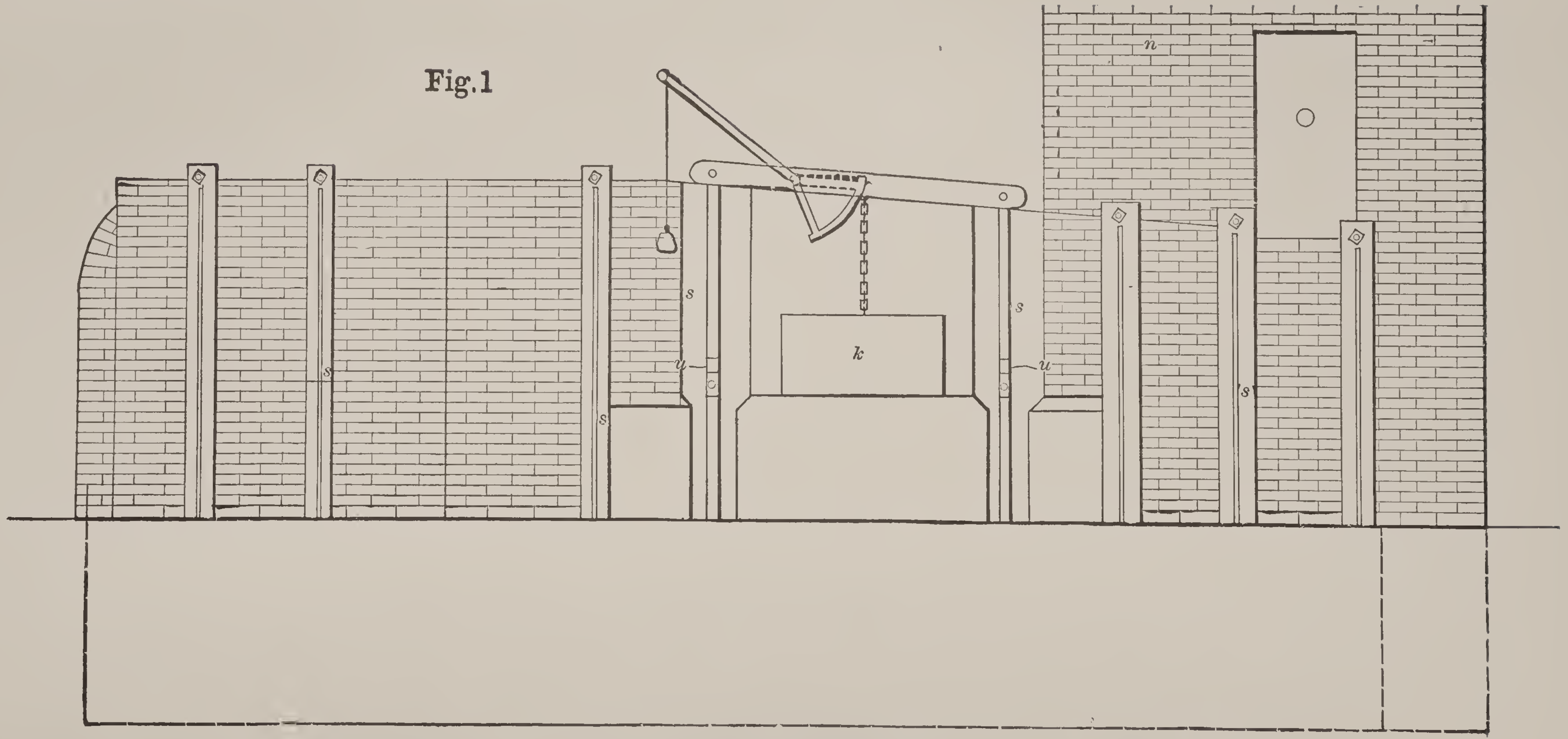
The capacity of a refining-furnace should be increased by deepening the hearth rather than enlarging its area, as the difficulty of retaining the copper in proper pitch is greatly heightened by increasing the surface area. Within certain limits, this may be effected by constructing a clay dam at the skimming-door; beyond this, the deepening must be effected by lowering the bottom, which, in any case, must pitch toward the ladle-hole from every point.

The size of the charge is limited rather by custom and the capacity of the attendants than by the size of the furnace, and has been greatly increased of late years.

During the writer's student years a charge of 14,000 pounds was considered large, but the present English refiners vary from 18,000 to 25,000 pounds and upwards.

The Lake Superior refineries are charged with some 18,000 pounds at 80 per cent. "mineral," producing over 14,000 pounds of pure copper. Of late years this charge has been increased 50 per cent. or more. This may be regarded as an average quantity; but the Orford Company, ever foremost in increase of capacity, has found no difficulty in refining charges of even 30,000 pounds. The principal trouble with large charges is the tendency of the refined copper to get "out of pitch," when retained in a molten condition for so long a time that fresh coal must be several times thrown upon the grate. Another difficulty is the want of room between hearth and roof to accommodate such a weight of metal. The shape and irregularity of the pigs of blister copper, and the difficulty of accurately placing such awkward bodies in the desired position in a red-hot furnace, have also prevented the ordinary use of larger charges. This is remedied in the Lake furnaces by lessening the customary pitch of the roof from bridge to charging-door, and curving it down abruptly at the latter point.

Fig. 1



REFINING-FURNACE AT THE LAKE SUPERIOR REFINING-WORKS.—SIDE ELEVATION.

Great care must be observed regarding the quality of all material allowed to enter the refining-furnace. It is not an apparatus for the concentration of matte, but simply to alter the shape of metal that is already nearly pure, and to put the finishing touches on it. Much of the pig-copper produced from blast-furnace work from both carbonate and sulphide ores may advantageously undergo a preliminary purifying process in the blister furnace. All copper below 96 per cent. should be thus treated; a mere melting down with free admission of air being sufficient to produce a 99 per cent. blister copper in most cases, so that two charges of 16,000 pounds each can be thus treated in twenty-four hours.

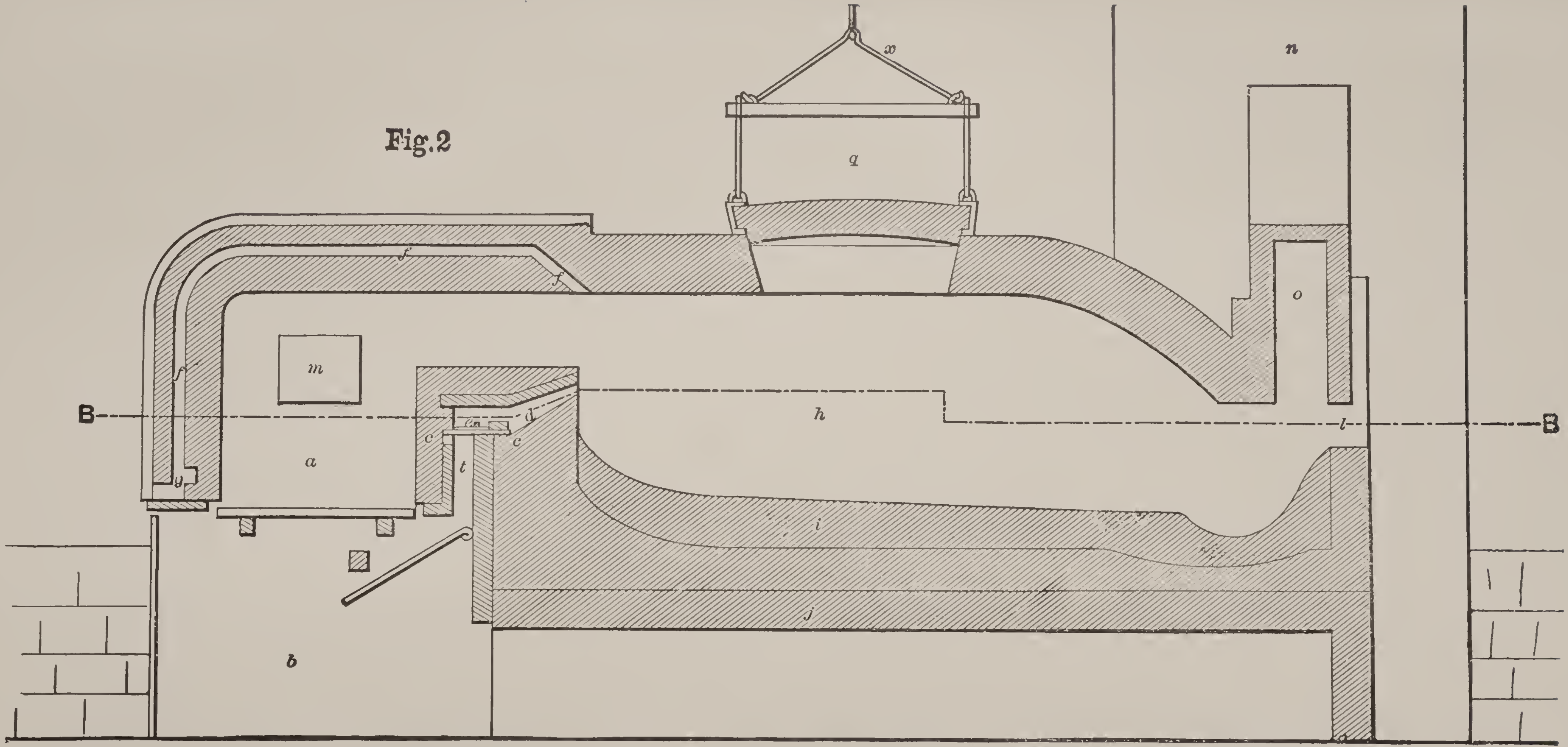
A few hundred pounds of the richest refinery slag from the last skimming may be returned to the same operation, the rest going back to the last preceding operation.

Cement copper from wet processes should, in most cases, be treated in the blister furnace. It must be thoroughly dampened to prevent mechanical loss, and when mixed with white metal to the extent of one-fourth or one-third of the entire charge, assists so materially in enriching the product and in shortening the operation that it just about repays the cost of its treatment.

Mr. James Douglas, Jr., has regularly produced so pure cement copper, by both the old and new Hunt & Douglas process, that it is refined at once with advantage. The principal drawback is its excessive bulk, which renders it necessary to add the cement copper in several successive portions, to obtain a full charge. This may be obviated by pressing it into bricks while yet damp.

Only one charge can be treated in a refining-furnace each twenty-four hours, and in ordinary cases, the labor connected therewith consists of one head refiner, three or four ladlers (according to whether the refiner acts also as ladler), one night refiner, one man to lift the ingots from the boshes, one to dump the molds, and one to remove any accidental impurities from the ladles, dry the molds, etc. The latter three operations are often conducted by boys. One man is also required to remove the ingots to the packing-house, while the packing itself, and the transportation and manipulation of the new charge, are effected by the furnace *personnel*, which usually expects to conclude the day's labor by three P.M. The work is very hot and severe while it lasts, and in the cases of the large charges referred to, extra

Fig.2



REFINING-FURNACE AT THE LAKE SUPERIOR REFINING-WORKS.—LONGITUDINAL SECTION.

assistance may be required for packing the copper and similar extraneous work.

While the quality of the copper depends largely upon the skill of the refiner, its external appearance and neatness are principally influenced by the ladlers. As these latter qualities exercise an undue influence upon the sale of copper in this country, it is of great importance to create a body of trained and skillful workmen, whose pride, as well as self-interest, is enlisted in the matter.

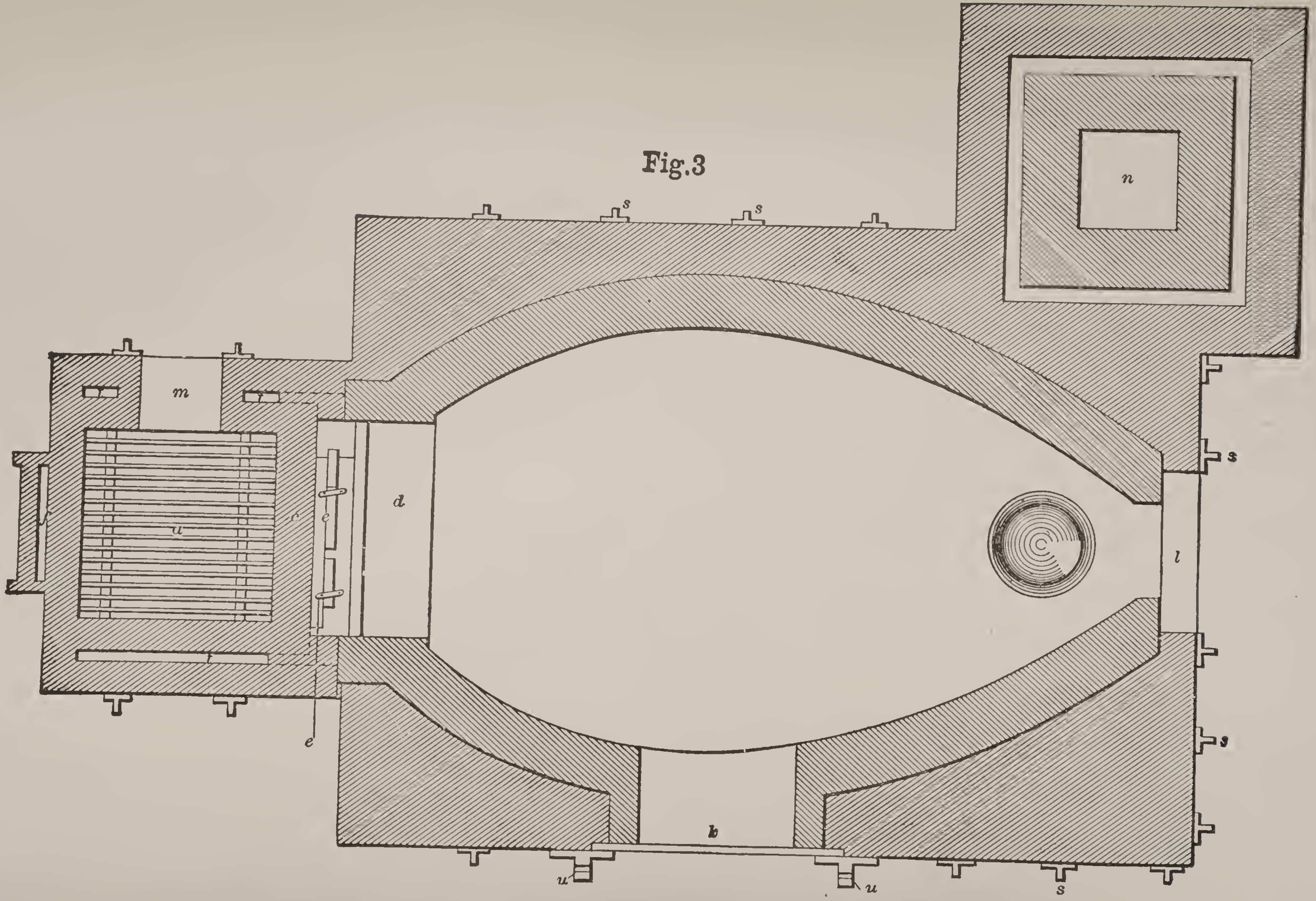
The color of the copper has an influence with American buyers entirely disproportionate to its importance as a sign of purity. A deep rose-red is the color most prized, while any brassy appearance is very damaging in the eye of the buyer.

As this dirty yellow appearance can be produced at pleasure by allowing the copper to remain a few seconds too long in the molds before it is dumped into water, while the poorest copper may be colored a fine, deep red by lifting it out of the water for a second immediately after dumping, and then returning it again to the trough, as well as by the use of any one of innumerable baths or "pickles," it certainly should not be regarded as of such vital importance. It is true, nevertheless, that *pure* coppers take on the desired color with much greater ease than those containing arsenic, antimony, or various other substances, and in the case of the remarkably pure Lake Superior metal, it is even difficult to produce that dreaded brassy appearance which any impurity of water or want of care is certain to develop in ordinary cases.

The red color is produced by the formation of a minute film of suboxide of copper; but why this hue should be affected by slightly brackish water, or by changes of temperature in the cooling water, it is difficult to understand. Pure water should be used and the most favorable temperature discovered for each variety of copper. In some cases, it must be nearly boiling; in others, ice-cold; while in still other instances, the refiner corrects an unfavorable coloring by the introduction into the cooling-boshes of soda-ash, salt, sawdust, coal ashes, and various other apparently inactive substances.

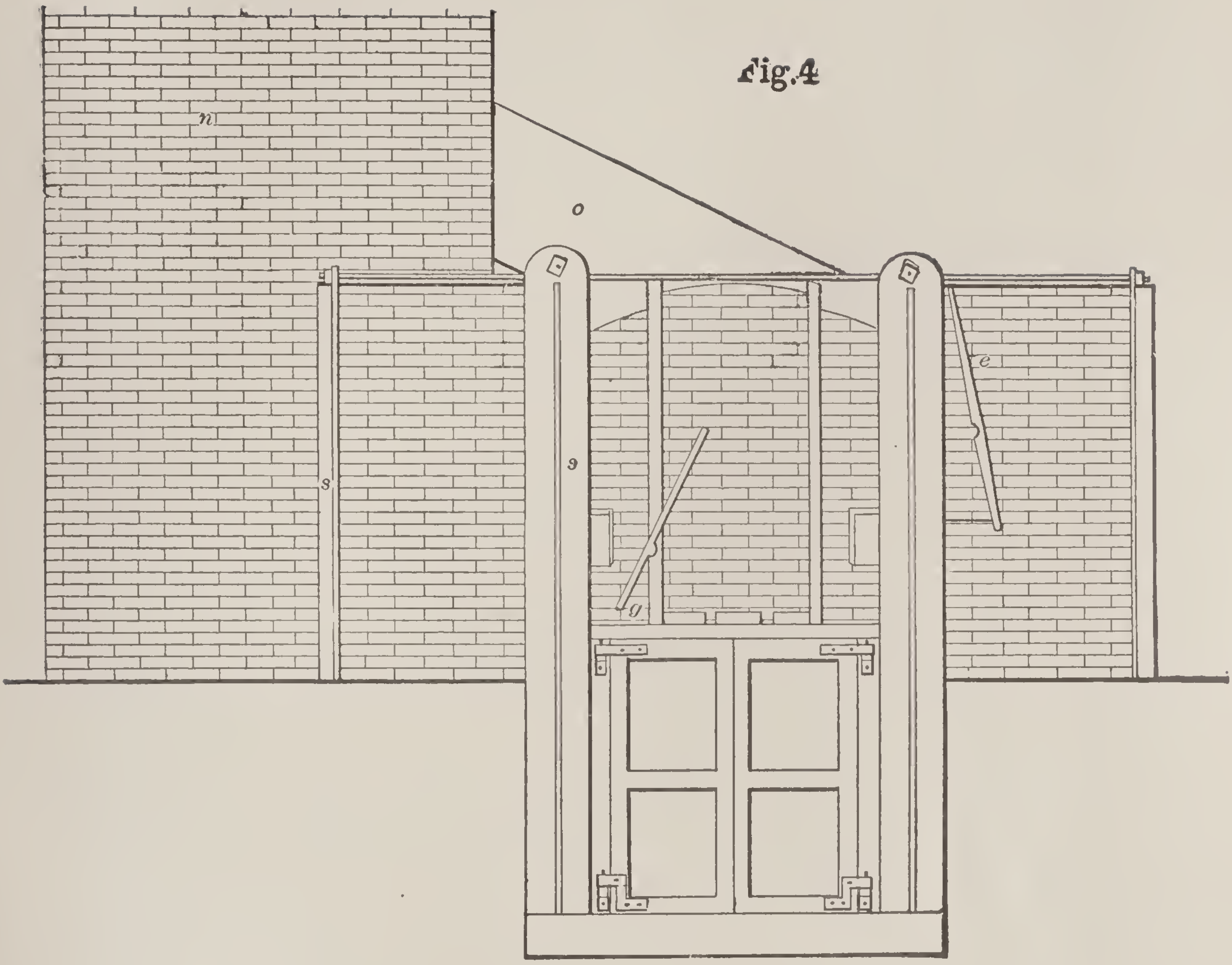
The refining-furnace usually receives its fresh charge immediately after it has been emptied of the preceding one. The fire is not urged until evening, in order that the first two stages of the operation, the *fusion* and the *refining*, may not be completed before

Fig.3



REFINING-FURNACE AT THE LAKE SUPERIOR REFINING-WORKS.—PLAN SECTION.

Fig.4



a	**Fire place.**
b	Ash-pit.
c	Bridge wall.
d	Air passage controlled by valve *e*.
e	Bridge valve.
f	Air passage controlled by the valve *g*.
g	Roof valve.
h	Laboratory of the furnace.
i	Hearth of sand & copper.
j	Arch supporting the hearth.
k	Charging door.
	Working door.
m	Fire door.
n	Chimney 58 feet high.
o	Flue to *n*.
q	Movable roof.
s	Cast Iron T bars holding furnace together.
t	Air passages leading to *d*.
u	Socket for bars *v* and *w*.
v	Bar for the repair spadel.
w	Pole bar.
x	Bar of movable roof.

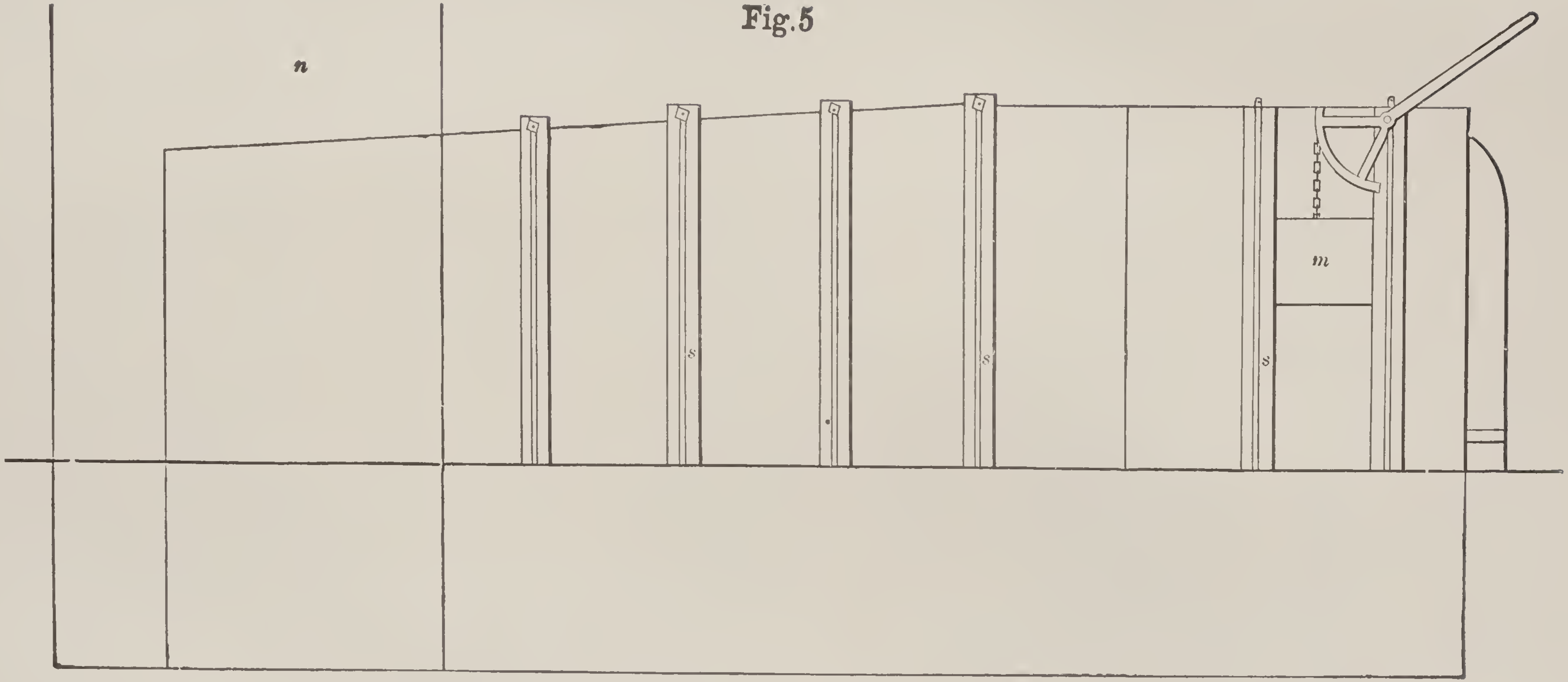
REFINING-FURNACE AT THE LAKE SUPERIOR REFINING-WORKS. REAR ELEVATION.—SCALE, $\frac{1}{4}'' = 1'$.

the day shift comes on to execute the *refining* proper—the third and final stage.

With pig-copper of reasonably good quality, the process of fusion may be begun at seven or eight P.M., and be pushed as rapidly as possible. Owing to the high heat-conducting quality of this metal, the pigs retain their shape until the fusing-point is reached, when they soften and melt almost instantaneously. From the reducing character of the flame, only slight chemical changes have thus far been produced; but as soon as the protecting layer of slag is removed from the surface of the bath, and air freely admitted, the process of purification proceeds with great rapidity, from the direct oxidation of the foreign substances present, as well as the more far-reaching and powerful reaction of oxide of copper upon all those metalloids and bases that have a greater affinity for oxygen than the copper itself. A thin slag forms rapidly upon the surface, and is removed at intervals of an hour or so. The constant escape of anhydrous sulphuric acid causes a persistent ebullition, which tends greatly to facilitate the process of oxidation. As the proportion of base metals becomes diminished, the slag is more strongly colored with the red oxide of copper, until that produced toward the close of this stage contains from 40 to 70 per cent. of this metal, and becomes a valuable oxidizing flux for the preceding blister process. The total amount of slag produced during the operation of refining depends principally upon the quality of the pig-copper, but is seldom less than 12 per cent. of the entire charge, containing from 4 to 6 per cent. of the total weight of copper.

The gradual cessation of ebullition and the rapid formation of oxide of copper by no means indicate the entire disappearance of the sulphur present, which, from its strong affinity to copper, remains dissolved in the bath with great tenacity. If the oxidizing process has been sufficiently thorough to insure the presence in the liquid metal of a perceptible quantity of suboxide of copper (from 0·2 to 0·7 per cent. according to different authorities), a small sample ingot poured at this stage will exhibit a very peculiar and characteristic phenomenon. On cooling, it will suddenly rise in a line along the center, often forming an abrupt ridge several lines in height, and having an irregular and granular fracture. This is said to be due to the absorption of sulphur-

Fig.5



REFINING-FURNACE AT THE LAKE SUPERIOR REFINING-WORKS.—SIDE ELEVATION.

ous acid, a property only possessed by metal containing a considerable proportion of suboxide of copper, but still unrefined and tenaciously holding on to a trace of sulphur and other impurities. The process of "flapping" or "rabbling" is now begun, by which the liquid bath, through the side door, is constantly agitated in a peculiar manner by means of a small rabble.

It is, of course, a purely oxidizing operation, both tedious and slow, requiring, on an average, two hours of constant work. Although seemingly a most awkward and ineffectual means of agitating an extensive bath of molten metal, and bringing all its particles in contact with the atmospheric air, it has never been improved upon. The copper now becomes "dry" from the dissolved suboxide, and when poured into a mold, sets with a deep depression upon its surface, while its fracture has a characteristic mottled appearance, following upon a previous fine-grained surface, as particularly mentioned by Professor Egleston in his valuable paper on *Copper Refining in the United States.* The color is a brick-red, but both grain and color are so influenced by the temperature at which the metal is poured, as well as by the rate of cooling as determined by the size of the test-ingot, that these signs must always be taken in conjunction with other and more reliable indications. The metal during this period is undergoing a powerful scorification from the dissolved oxide of copper, and most injurious impurities are gradually oxidized, and either effectually removed by slagging or volatilization. Certain metalloids, however, resist this scorifying influence to a remarkable degree, and consequently have a most injurious effect upon the refined metal. These are arsenic, antimony, and tellurium, mentioned in the order of their frequency. The extreme importance of the subject warrants the mentioning of the best means to remove the two first-mentioned impurities, the latter having come but once within the author's experience, and probably requiring the employment of one of the electric or chemical methods, by which excellent copper can be made from very poor material.

A careful trial of Vivian's invention of dry-sweating, by which the impure blister copper is exposed to a long oxidizing heating just below the fusion-point, has not succeeded with the writer; but the addition of from 3 to 5 per cent. of pure white metal—subsulphide of copper—to the bath at the beginning of

Fig.1

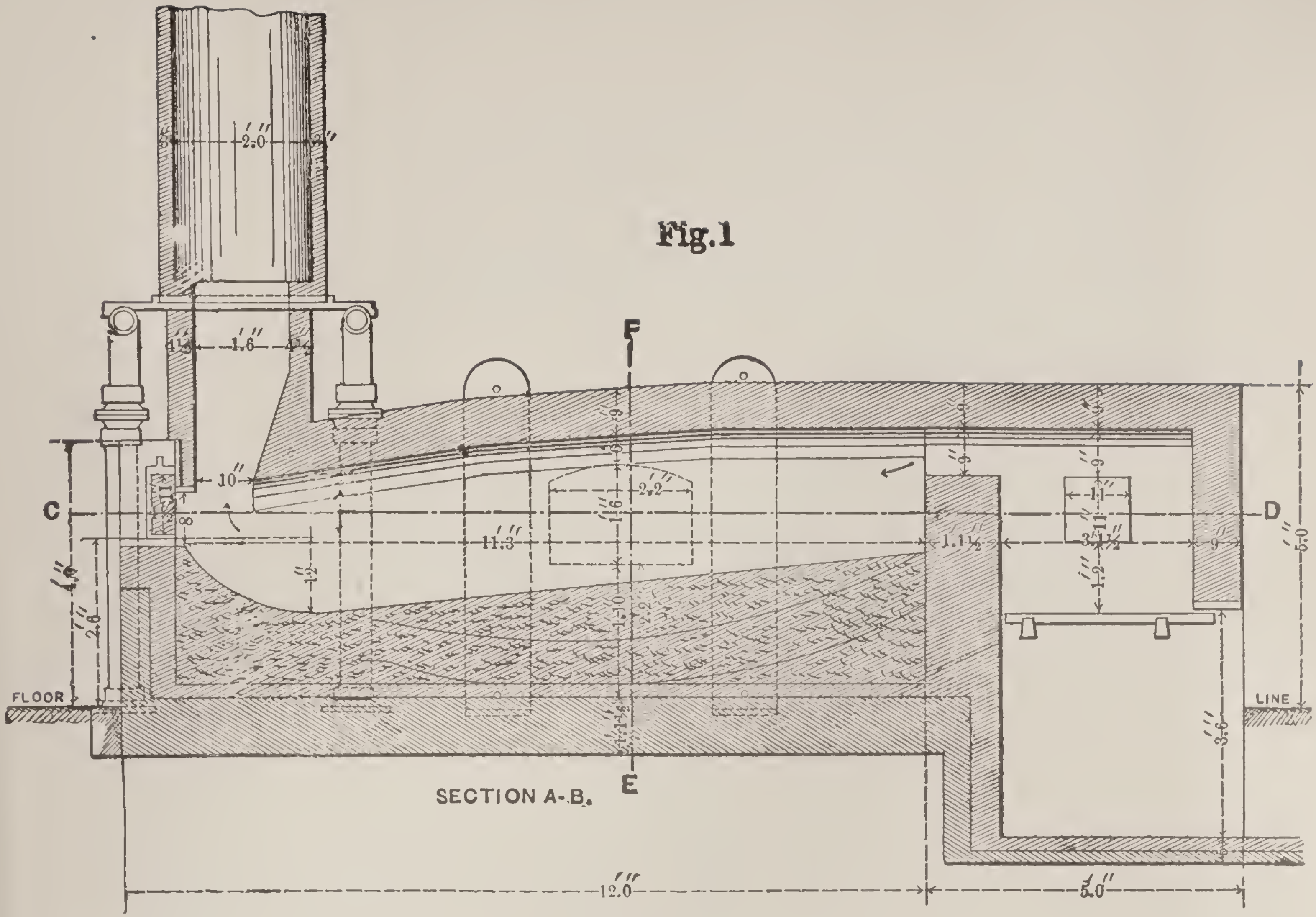
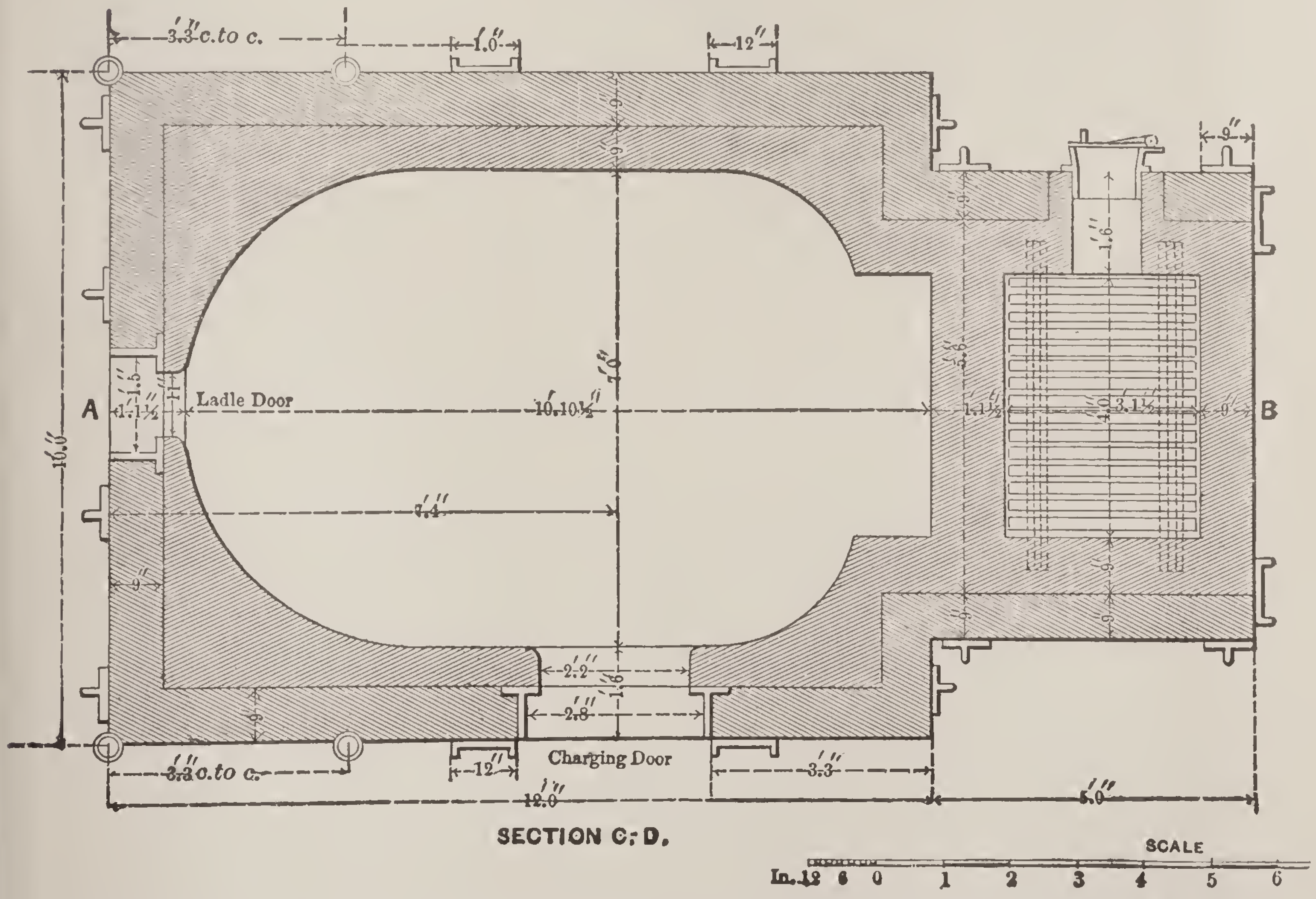


Fig.2



PITTSBURG COPPER-REFINING FURNACE.—FIG. 1. LONGITUDINAL SECTION. FIG. 2. SECTIONAL PLAN.

the refining process (as suggested by some person forgotten by the author) has a most rapid and satisfactory effect in removing both arsenic and antimony. Very bad cases may require two such additions, with an intervening oxidizing operation. A still more sure and radical method consists in exposing the arsenical ore to a dead roast, and subsequently smelting the same with a large proportion of iron pyrites—cupriferous, if possible. The resulting low-grade matte should be regarded and treated as a sulphide ore, and will, if the initial calcination is thoroughly conducted, be free from either arsenic or antimony.*

The process of reduction follows that of oxidation, and the suboxide of copper, having served its purpose as a purifying agent, must now be reduced to metal again; otherwise, the copper would be brittle both when cold or at higher temperatures, and unfit for manufacturing purposes. The reduction is effected by means of a long pole, as large as can be introduced into the furnace and of any kind of green wood—hard wood being the most economical. This, being buried in the metal bath, evolves an immense volume of hydrocarbons and other reducing gases, and rapidly removes the excess of oxygen. The surface of the metal is also covered with charcoal, to prevent access of air, and samples are constantly taken to determine the condition of the copper. The entire removal of all the oxygen present is impossible, even over-poled copper, according to Egleston,† containing over 0·1 per cent. of oxygen. And otherwise tough copper may become brittle from over-poling, and this is doubtless due to the fact that the impurities that were present in the tough copper were dissolved as oxides and consequently innocuous, but on being reduced to the metallic state, at once asserted their deleterious influence.

The poling usually lasts an hour or more, and is continued until a full-sized test ingot shows no contraction or depression on cooling, and the texture is extremely fibrous and silky, and of a beautiful rose-red. Further tests are made by nicking and

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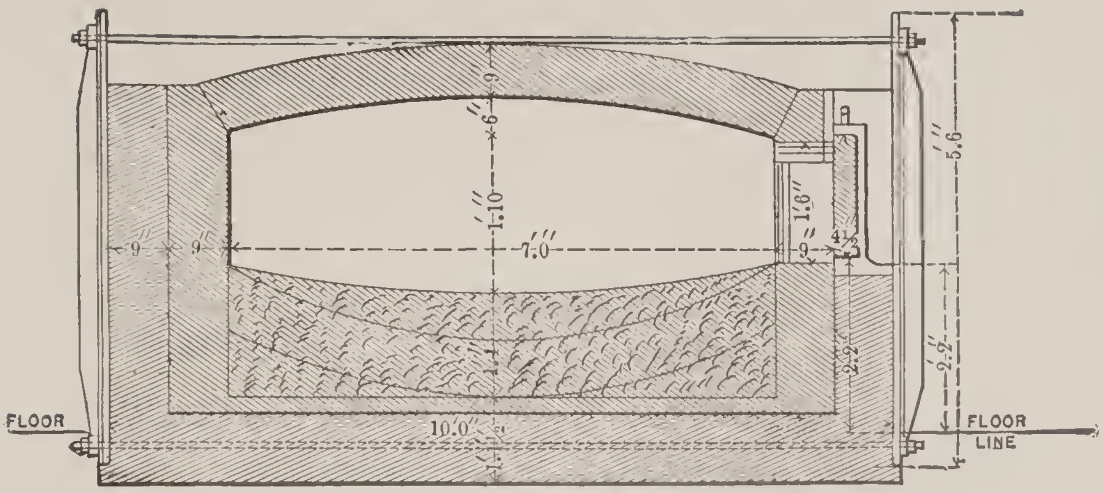
* The author is unable to give the original sources of many statements here made and tested by himself with satisfaction, and desires to distinctly disclaim any originality in any operation or apparatus pertaining to copper metallurgy; having always preferred to adopt those improvements that have been thoroughly tested by others of a more original turn of mind.

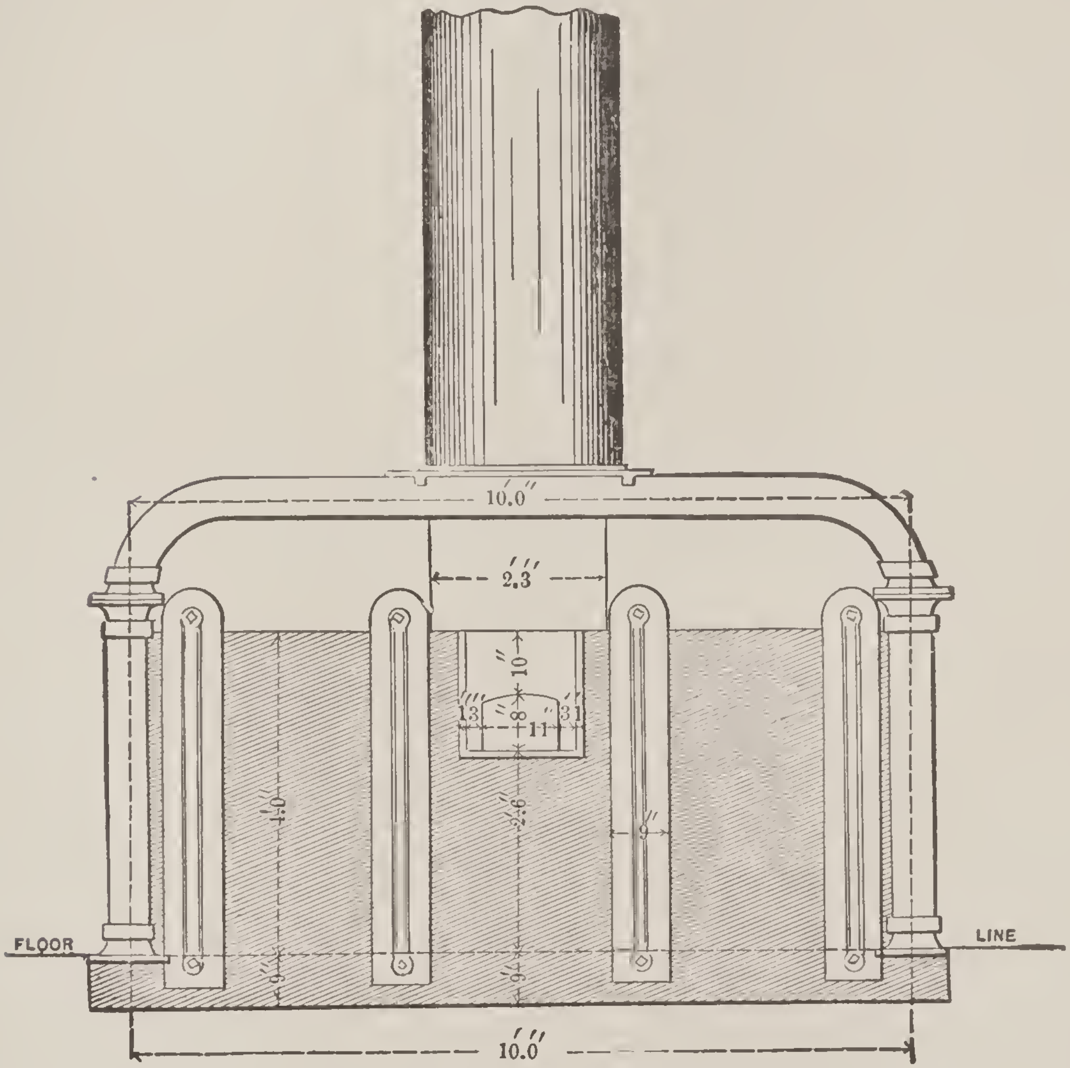
† See *Copper Refining at Lake Superior*, by T. Egleston.

Fig.3



SECTION E.-F.

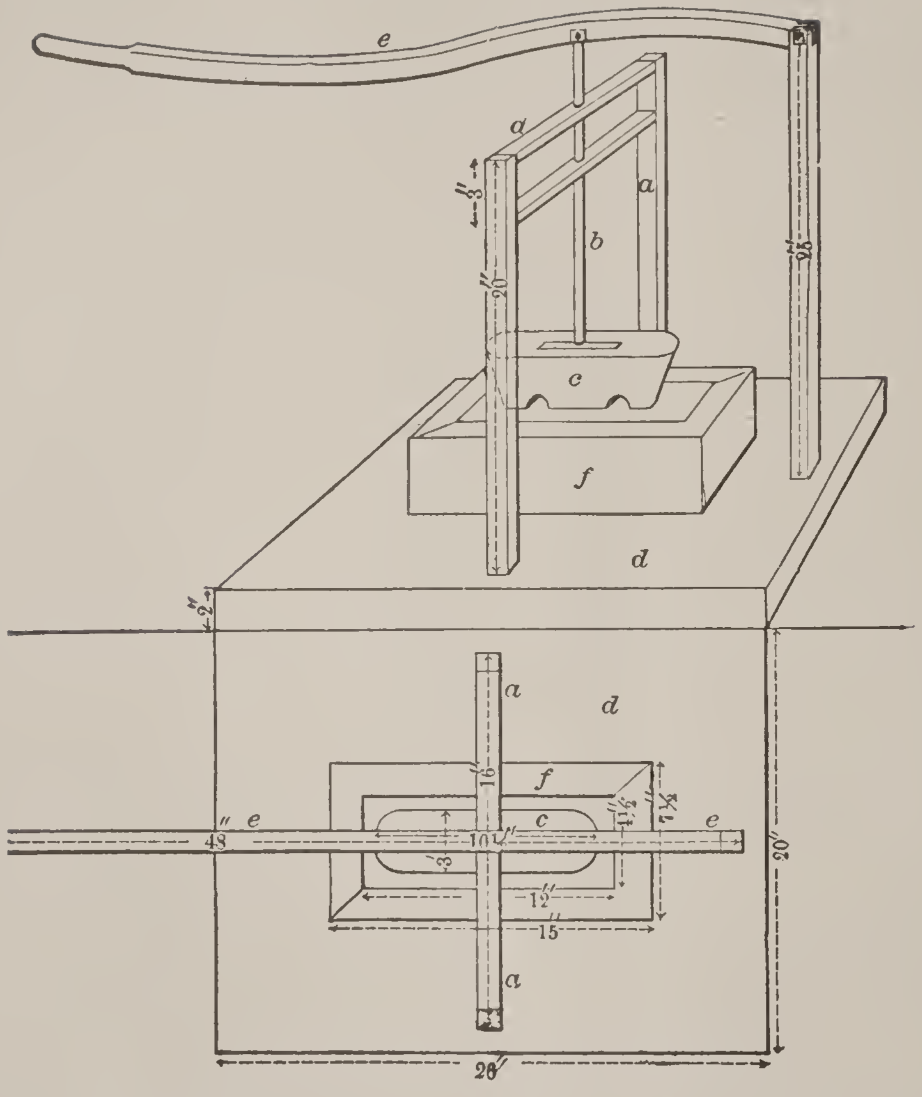
Fig.4



FRONT ELEVATION

PITTSBURG COPPER-REFINING FURNACE.—FIG. 3. CROSS SECTION. FIG. 4. FRONT ELEVATION.

Fig.5



PITTSBURG COPPER-REFINING FURNACE.—STAMP FOR INGOT MOLD.

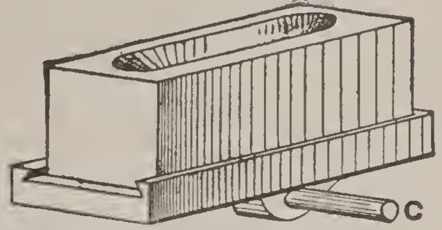
bending test bars, and by hammering out a piece into a thin plate, which should show no cracks at the edge. This condition of tough-pitch is essential to copper used for rolling or wire drawing, but is entirely superfluous for ingot copper that is to be used for brass founding, as it may be easily imagined that the fusion that it undergoes in the brass-founder's crucible, under various oxidizing and reducing influences, effectually upsets the exquisite niceties of the refining process, so far as the proportion of dissolved suboxide is concerned.

A volume could be easily filled with practical comments upon the process of refining, but space forbids any further details. The addition of lead to copper intended for rolling is quite common in England, and is doubtless beneficial to many impure coppers. The purer copper of the Lake district and from the Arizona carbonates does not seem to receive any benefit from this practice.

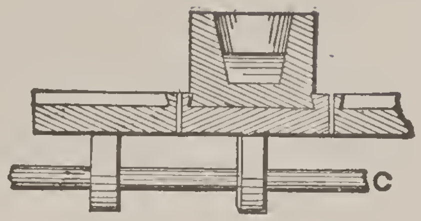
The molds used for the casting of ingots should always be made of copper, and are easily and rapidly produced by the ordinary ingot stamp, as illustrated herewith. The proper taper of the mold and the proportion of surface in contact with the ingot have an important effect upon the ease with which the mold delivers. When the copper is ladled too hot, the molds are rapidly ruined, and as at best they wear rapidly, they should be returned to the refining-furnace as fast as they become in the least imperfect; otherwise, constant annoyance and accidents will result from the obstinate sticking of the ingots.

The ladles used in the refining process come almost exclusively from England, and are made of a peculiar quality of iron. They last from 10 to 100 operations, according to the temperature of the copper and the care bestowed upon them.

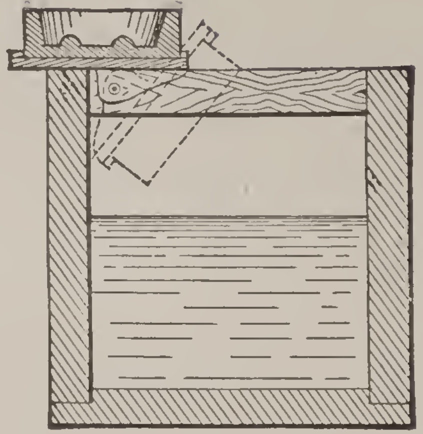
The Ansonia Brass and Copper Company has patented a new mold for casting ingots directly from the furnace, without the intervening process of ladling. While such an improvement would relieve the workmen from the hot test and most laborious portion of the operation, the very nature of the metal, its high fusion-point and great heat-conducting capacity cause it to chill so suddenly as to render the success of such an invention a matter of some doubt. The same company employs a gas generator for heating a single refining-furnace, and although pronounced convenient and successful, it can hardly make any great saving,

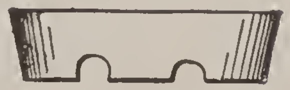


PERSPECTIVE VIEW OF SHOE AND MOLD.

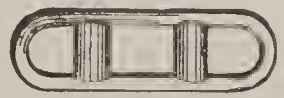


SECTION B——B.



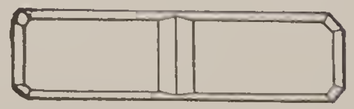


SIDE OF INGOT.

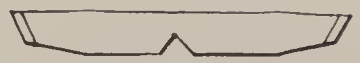


BOTTOM OF INGOT.

ENGLISH COPPER INGOT.



BOTTOM VIEW OF INGOT.



SIDE VIEW OF INGOT.

DETAILS OF INGOT, INGOT MOLD, SHOE AND TANK.

considering the small amount of fuel generally used in ordinary refining, and the great expense of the generator plant.

A great saving in the expense of refining has already been made by increasing the capacity of the ordinary furnace, and the next important improvement may be looked for in bettering the quality of the refined copper and increasing its strength and tenacity. How this is to be effected is far too difficult a subject to be discussed within the limits of a practical paper on existing methods. Experiments conducted by Mr. Patch, of the Detroit Copper Company, as well as the writer's personal trials, seem to indicate that the presence of suboxide of copper is by no means essential to the greatest malleability and strength, as believed by Percy, and that a proper method of treatment may result in the production of copper having a strength far beyond the best brands at present known.

The cost of refining varies so greatly with the purity of the blister copper treated, and depends also so completely upon the size of the charge, that no absolute estimate of expense can be given.

The following figures, taken from actual practice, give a fair idea of the cost of refining ordinary Arizona pig-copper of from 95 to 98 per cent.—being about equivalent to good Chili bars. The size of the charge is assumed to be sufficient to produce 24,000 pounds of refined metal, the furnace running regularly, and making one charge every twenty-four hours, while the expense of foremen, etc., is supposed to be divided between two furnaces.

Cost of refining one charge, yielding 24,000 pounds of copper—

Coal—best quality—3·8 tons, at $5.50	$20.90
Clay and sand for fettling—450 pounds	.95
Cheap clay and loam for doors and slag-beds—400 pounds	.30
Poles—45 feet of 6-inch poles, at 4 cts	1.80
Charcoal—6 bushels, at 10 cts	.60
Proportion of cost of renewing bottom	.34
" " " main arch	.62
" " " flue	.32
" " other repairs on furnace	.72
For renewing tools, barrows, ladles, etc	1.11
Repairs on ditto	.64
Carried forward	$28.30

Brought forward	$28.30
Lights, oil, soap, clay-wash, brushes, etc	.70
Cost of resmelting poorer slag in blister furnace	1.20
One head refiner	4.00
One night refiner	3.00
Four ladlers, at $2.75	11.00
Man fishing ingots	1.50
Boy dumping molds	.75
Boy removing specks from ladles while pouring	.75
Man wheeling copper to packing-room	1.50
One laborer about furnace	1.50
One head packer	2.50
Two assistants, at $1.50	3.00
Miscellaneous expenses of packing, paint, stencils, etc	.65
Cost of pumping water for boshes	1.15
Proportion of day and night foreman	3.00
Proportion of expense during Sundays, and other delays.	1.44
Proportion of assaying necessary for control of operation.	1.12
Grand total	$67.06

Cost per pound, ·2794 cts.

This agrees closely with the actual cost of running large refining-works where prices closely approximated those assumed in this estimate; being just three-tenths of a cent, including general expenses.

CHAPTER XV.

REFINING COPPER WITH GAS IN SWEDEN.

At Atvidaberg,* in Sweden, a regenerative gas plant is used to heat the small reverberatory furnace in which the cement copper, obtained from cupriferous pyrites by the use of the Henderson extraction method, is first made into blister copper, and eventually refined upon the same hearth, though in a second operation.

This plan of using the same furnace for the making of both blister copper and refined copper can by no means be recommended, as the impurities which remain within the furnace from the first operation are sure to hinder the more delicate end-operation. But where the product is small, as at Atvidaberg, it is sometimes necessary to work with imperfect apparatus, and the small proportion of sulphur in the cement copper removes the principal objections to this plan of procedure.

The small scale of working here practiced cannot be recommended, as it is neither conducive to economy nor to a high grade of product; but the successful employment of a gas-producer in copper-refining is so interesting and instructive, and bears so strongly upon the conditions existing in many parts of our own country, that I think it well worth while to embrace the opportunity that has presented itself of describing it minutely, and with detailed working-drawings to illustrate the apparatus employed.

The cement copper is charged in a moist state, containing 50 to 60 per cent. copper, with refinery-slags and a small amount of roasted, high-grade matte, and a considerable proportion of charcoal, to prevent the oxidation of the fine particles of metallic copper.

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* Mr. Paul Johnson, of Sweden, an experienced copper-metallurgist, has kindly furnished this description of work at Atvidaberg.

A low-grade blister copper is produced herefrom, carrying only some 92 per cent. copper, and thus far inferior to any blister copper that is produced in this country in a reverberatory furnace. The grade of this copper could be easily improved by lengthening the period of oxidation. But it has been found more economical to throw more work on to the refining part of the operation, and produce a lesser proportion of rich slags from the preparatory process. The quality of the refined copper is entirely satisfactory, as indeed it ought to be, considering the favorable composition of the ores from which it is made, they being mostly cupriferous iron-pyrites, which is first burned in kilns to utilize the sulphur contents for acid manufacture.

The gas generator is fired entirely with a very cheap and poor quality of fuel, consisting of refuse wood from building operations, ends of scantling, slabs, limbs of trees made into fagots, and similar waste material, and the gas is purified from tar by direct contact with water.

THE BLISTER PROCESS.

The operations here described were executed in the year 1889, and starting with a cold gas-producer, and furnace hearth, the latter merely having been slightly warmed by a small wood fire upon the hearth.

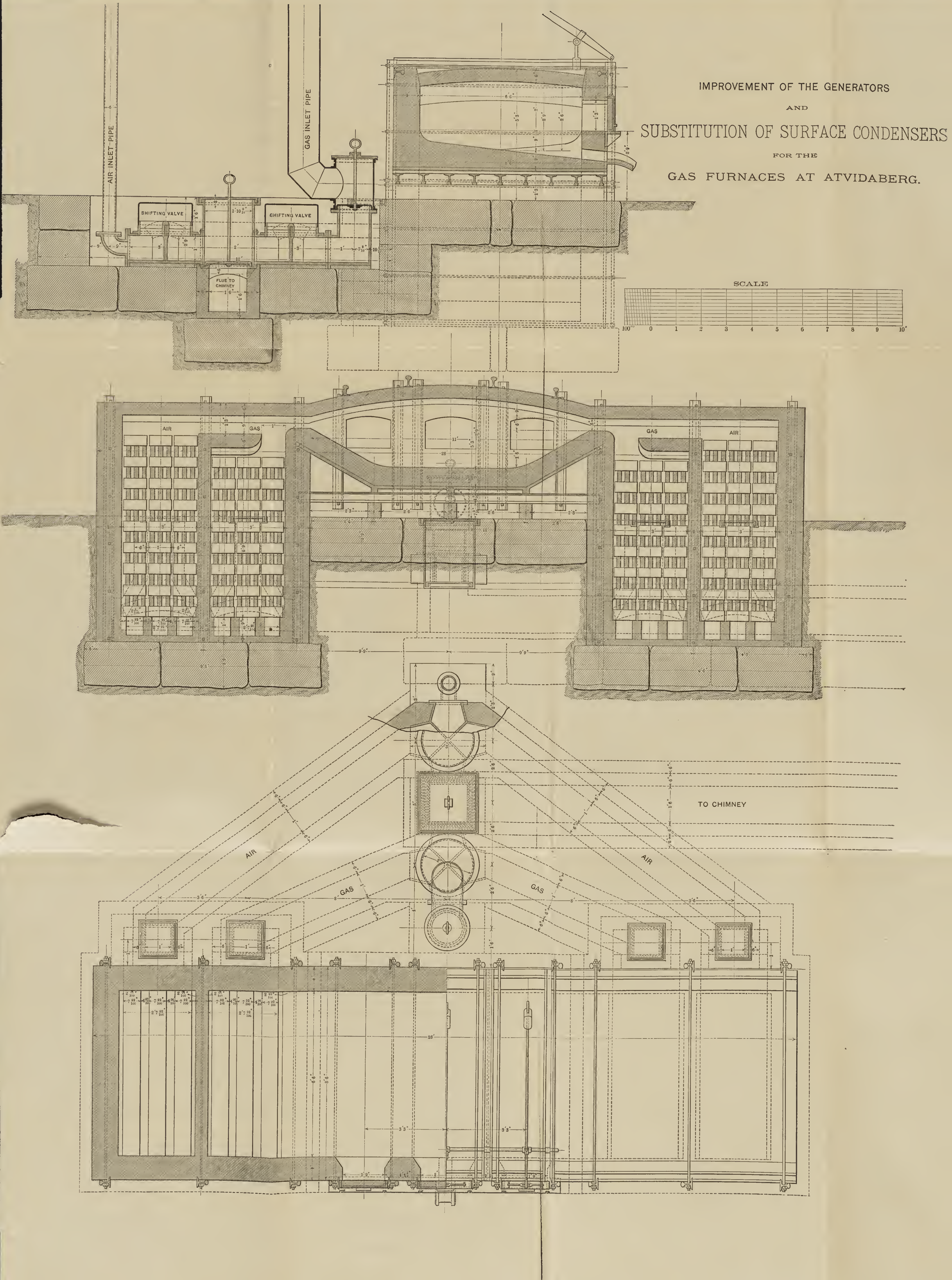
This small fire was kept upon the hearth about sixteen hours before filling up the gas-producers, the flame being allowed to pass out through the regenerators on either side.

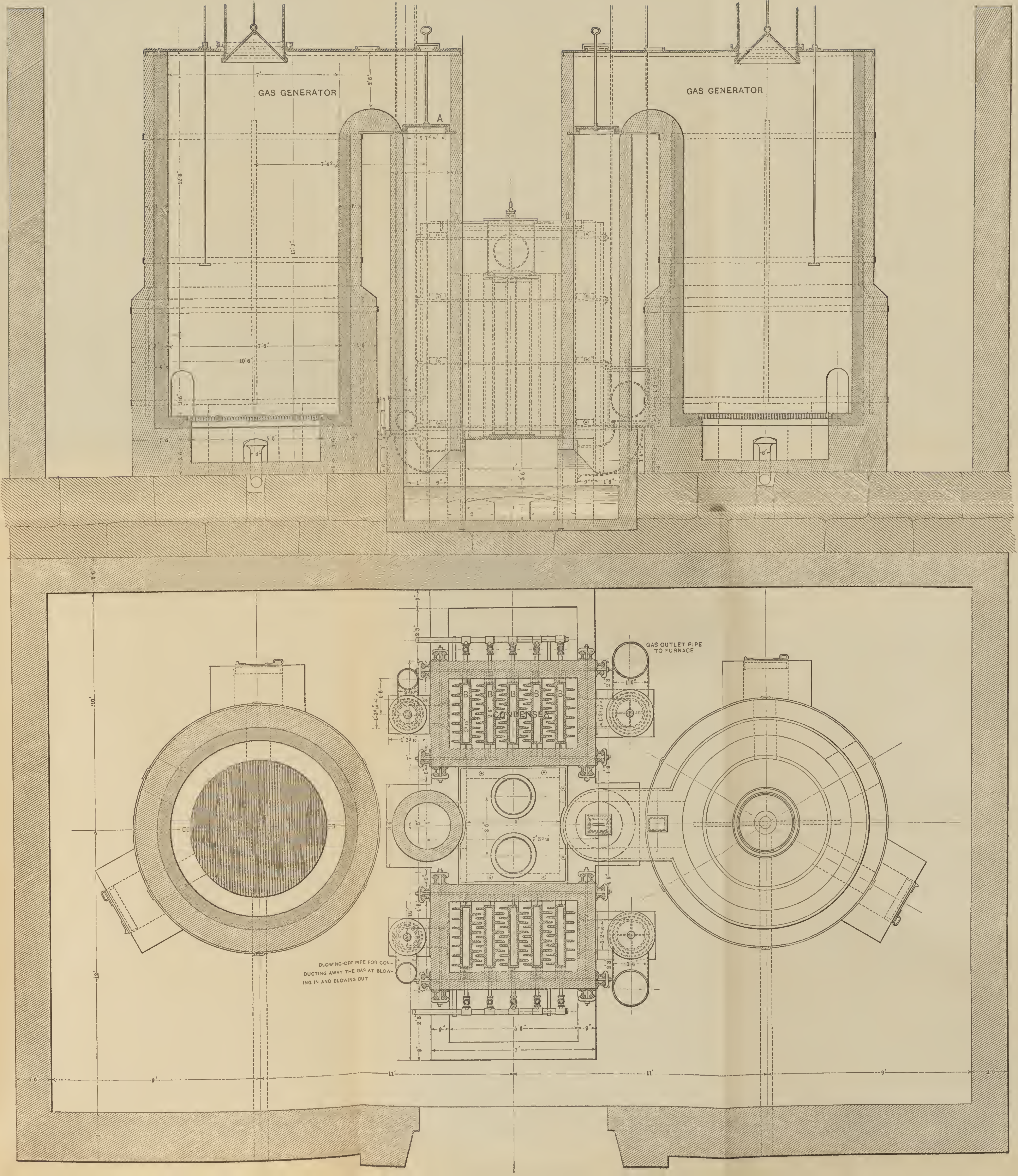
At the expiration of this time, and at about 6 A.M., the fire in the gas-generator was kindled, the latter within the next three hours being filled nearly to the top with the refuse wood already mentioned.

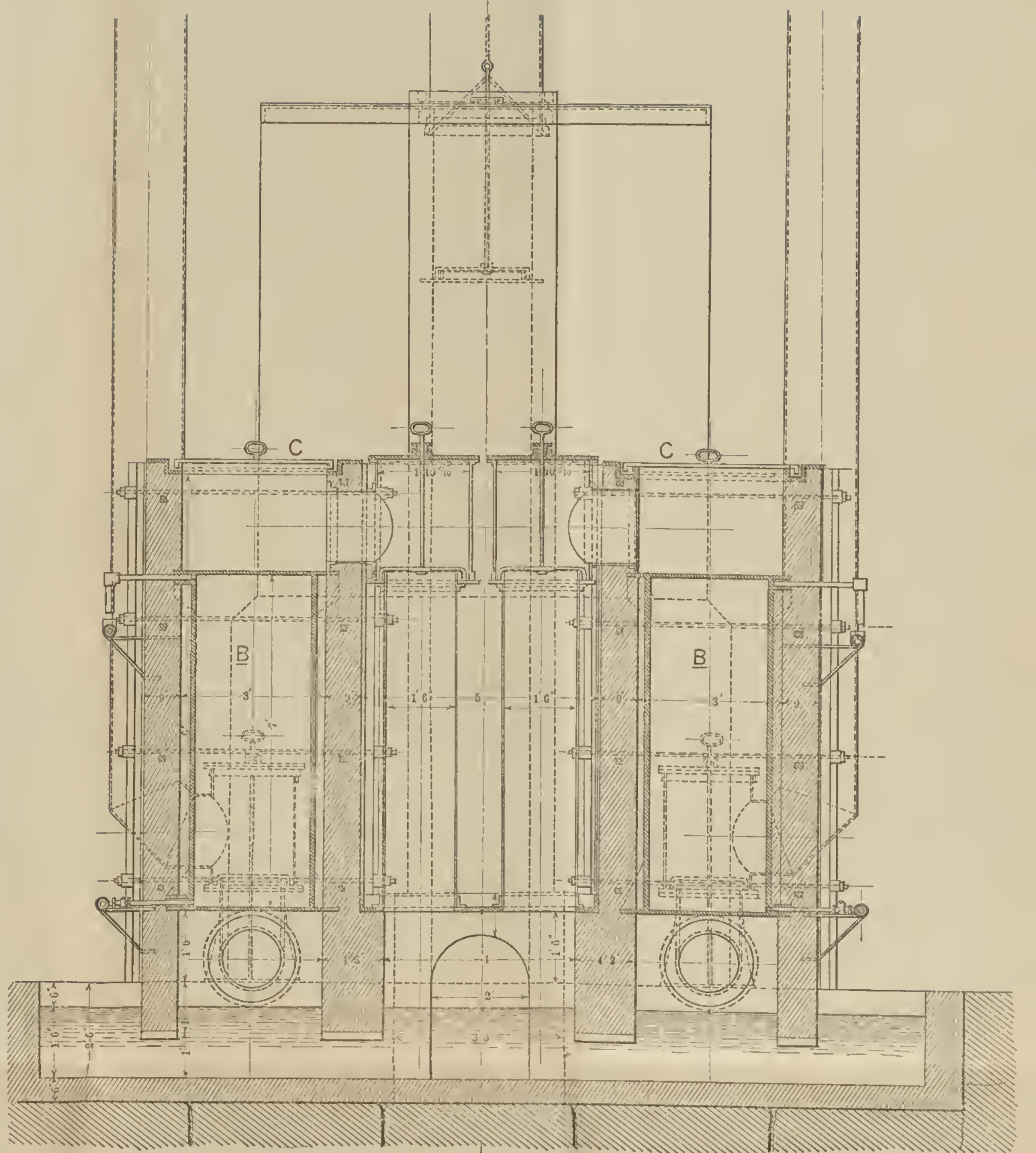
By 10 A.M., four hours after the kindling of the fire in the generator, it was considered safe to let the gas into the furnace, it having hitherto passed off into the air. Any carelessness in letting the gas into the furnace before all air is driven out of the generator might result in a serious explosion. The gas is shifted from one regenerator to the other every 10 minutes by means of a valve, and by 6 : 30 P.M., 12½ hours from starting, the hearth of the furnace has reached a bright yellow heat.

The bridge-wall and other exposed parts were then repaired in the customary manner by tamping into the burnt-out cavities

IMPROVEMENT OF THE GENERATORS
AND
SUBSTITUTION OF SURFACE CONDENSERS
FOR THE
GAS FURNACES AT ATVIDABERG.




GAS GENERATOR
GAS GENERATOR
A
B
B
B
B
B
CONDENSER
GAS OUTLET PIPE TO FURNACE
BLOWING-OFF PIPE FOR CONDUCTING AWAY THE GAS AT BLOWING IN AND BLOWING OUT
7'
7' 6"
10' 6"
12' 3"
5' 6"
5' 6"
7'
9'
11'
11'
9'
10'
12'

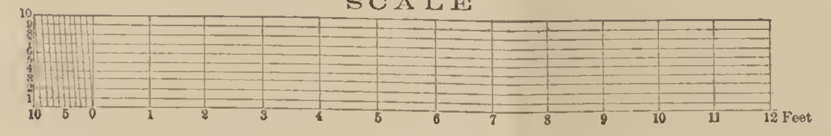


GENERATORS AND CONDENSERS

—FOR—

ATVIDABERG GAS FURNACES.

SCALE



crushed quartz, sieved through a screen with eight meshes to the linear inch, and at 8 : 15 P.M., 14¼ hours from the start, the furnace was hot enough to begin charging.

Nine cubic feet of charcoal fines (a totally valueless material where charcoal is used, and that can be obtained without cost) was first dumped upon the furnace-hearth, then 300 pounds of thoroughly roasted copper matte containing about 50 per cent. copper, then 600 pounds of refinery slags, containing 40 to 60 per cent. copper, then some two cubic feet of charcoal again; on top of this came 1,650 pounds of cement copper, and last of all, 300 pounds more of roasted matte, after which the doors were luted and the gas turned on again.

The great bulkiness of the cement copper requires that, in this small furnace, the charging should be done in two separate operations. At 9 : 45 A.M., 13¼ hours after charging, the material was sufficiently fused down to admit the addition of the remainder of the charge. Four cubic feet of charcoal fines were first thrown upon the half-molten bath, then 1,500 pounds cement copper, then 300 pounds each of roasted matte and refinery slags, and, last of all, about 40 pounds of dirty cement copper, from the straw filters of the precipitating tanks.

The working-door was luted up at 10 : 05 A.M., and at 2 P.M., after four hours' firing, the charge was sufficiently fused to admit of the first slag-skimming, which was conducted in the customary manner. At 3 : 30 and 4 : 45 P.M., slag was again skimmed in small quantities, the charge still being far from liquid all through, and much of it still sticking to the hearth.

As the blister copper is here tapped into iron molds (an excellent plan which cannot be too highly recommended to our own smelters for its cleanliness and economy), these are now heated with slag from a fourth skimming at 5 : 45 P.M., at which time it was found that the bottom was getting clean, and that bubbles were beginning to rise through the molten mass, showing that the chemical reactions between the sulphides and oxides present, as well as between the carbon (charcoal) and oxides were rapidly progressing. At 6 : 20 P.M. a considerable quantity of tolerably liquid slag was skimmed, and a thorough stirring of the bath with a stout iron hook took place, to free the bottom and assist in breaking up the still unfused masses. A slight blast was now forced through openings in the fire-bridge to cool the overheated brick-work in that portion of the furnace, and at 6 : 40 P.M. slag was again skimmed. At 7 P.M. the surface of the copper was skimmed tolerably clean of slag, though a considerable film of matte still floated upon it, and the blast was now let on through two blow-holes in the rear of the furnace to hasten the oxidation of the sulphur and iron. Sometimes, instead of using blow-holes, a tuyere is introduced into a small door opposite the charging-door, and there luted in place.

At the same time the admission of air to the gas-producer was increased so as to make the flame more oxidizing, and permit it to play more freely over the whole extent of the bath.

Between 7 and 9 : 20 P.M., four skimmings were made, and at the last one, the bath was again skimmed clean, and a test taken in a small ladle, which showed a pretty even surface.

The sulphurous acid gas is escaping rapidly in a multitude of fine bubbles (German "Braten"), and this action increases in violence until globules of molten copper are thrown out through the working-door. The slag, which at first was dense, now becomes pasty and full of bubbles.

Tests taken every few minutes showed a disposition to rapidly become thinner and more concave, until at 10 : 10 P.M. they assumed their proper form, *i.e.*, swelled up with a hollow in the center like a baker's roll, the bottom portion, when broken, showing a reticulated surface.

The blister copper is now tapped at once, after a few moments of strong heating, into iron molds, formed on the bottom by a large iron plate, while the sides are made of strips of iron shaped like a triangular gutter, and laid on their open side, the apex of the triangle being uppermost. Small cross-divisions of sand are made between these parallel sides, and the sand is covered with a thin layer of clay mortar to strengthen them, and which must be thoroughly dried before the metal is tapped into them.

The tapping of the blister copper is accompanied by a strong evolution of sulphurous and sulphuric-acid gases, and requires some skill in its management, being guided and controlled in its impetuous course by long wooden poles in the hands of the furnace-men. As in the common English and American practice, the pigs are broken apart at their junction while still almost white-hot, and as soon as the outside of the pig has become strong enough to allow it to be pried upon with a heavy iron bar.

At this stage, the blister copper is very red-short and brittle, but if the critical moment is neglected, it becomes tough and unmanageable, necessitating a most laborious and expensive recourse to cutting apart with chisels. A little sand or fine charcoal is thrown upon the necks of the pigs to keep them hot until the pig has solidified enough to bear handling.

The furnace is now scraped out clean, and the sides and bridge mended with crushed quartz in the usual manner, a long plug of wood being first driven into the tap-hole, and thoroughly packed with damp sand, so that it soon carbonizes, and becomes a plug of charcoal which is easily penetrated by the tapping-bar. On this small furnace, about 140 pounds of quartz are used for repairs after each charge, and this is heated strongly for about an hour.

Only one charge is made in twenty-four hours, the actual time taken for the operation being about 21½ hours, and the furnace-gang consisting of one head smelter, one assistant, and one man at the generator, all on eight-hour shifts.

The consumption of fuel in this operation is very difficult to estimate, as every scrap of waste wood is used, in the shape of old roots, branches, ends of sawn-off timbers, etc.; also sawdust. About 850 cubic feet of waste limbs and 400 cubic feet of waste wood were used in twenty-four hours, but as this was all estimated by measure, and as from its irregular shape, the interstices bore an enormous proportion to the amount of solid wood employed, it is evident that a very reasonable consumption of fuel took place.

The taking of tests from the bath, and the judging of the forwardness of the operation therefrom, are difficult and delicate, and depend largely upon experience. As the blister copper in our large blister-furnaces presents exactly the same series of changes, and as inexperienced men are constantly perplexed and mystified by this matter of tests, I will give a brief description of the appearance of these tests at Atvidaberg, as the same description will apply equally well to our own conditions.

The tests are taken at most works in a small ladle about 1½ inches in diameter by ½ inch deep.

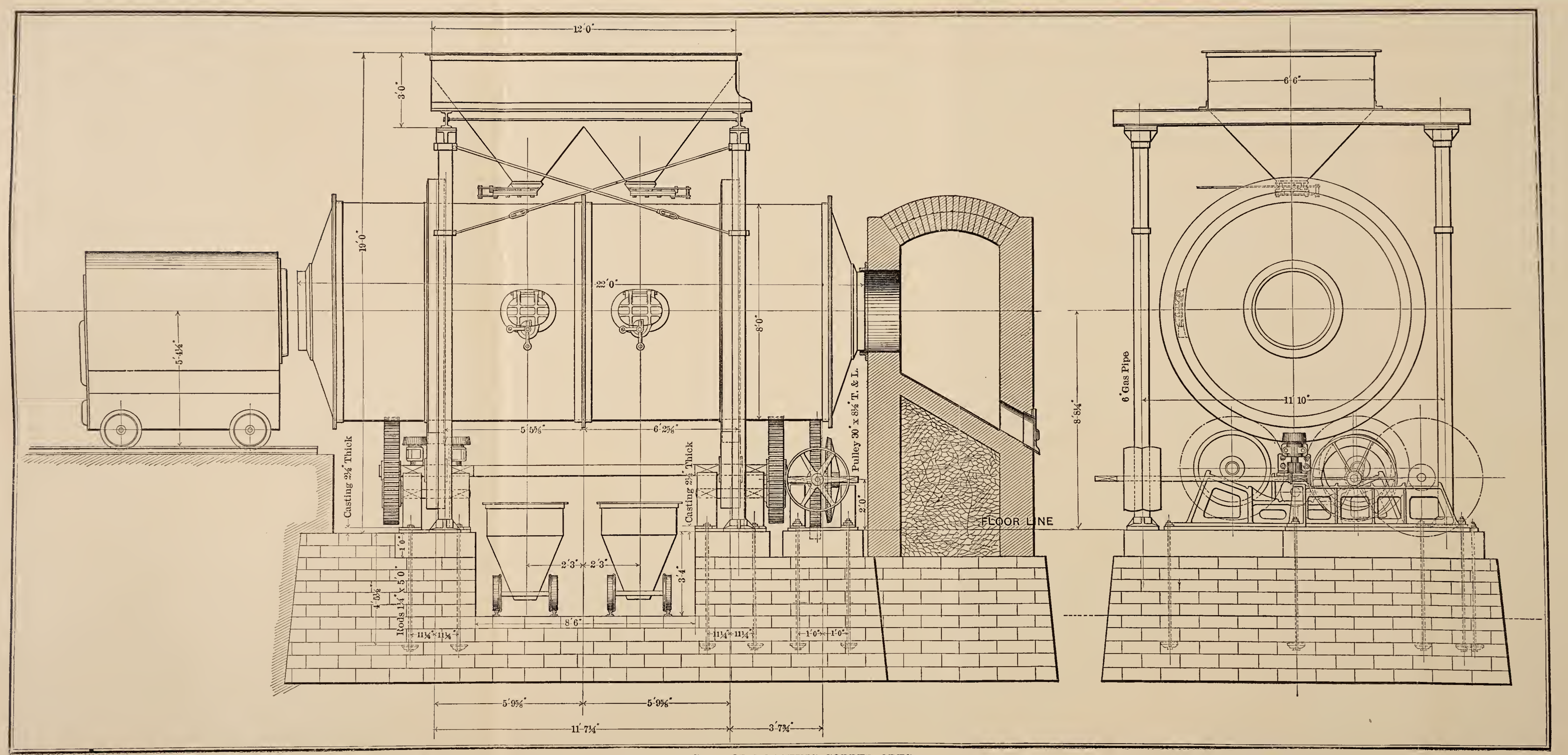
A test taken early in the operation, and before the process of oxidation has advanced very far, is level at surface, and shows a dull gray fracture, with few shining points.

As the oxidation of the sulphur progresses, the assay seems to spread out thinner, and on fracture shows less dullness, and a multitude of small bubbles begins to appear.

Still farther on, the bubbles begin to enlarge, becoming the size of small peas, while the surface of the test rises and becomes more convex. This increases till it looks very much like a baker's roll, and shows an indescribable vitreous luster in the interior of the bubbles on fracture, when the blister process is complete, and the copper is ready to tap. If the oxidation were carried further, there would not only be a heavy loss of copper in the slag (there is a much less loss in the refining operation later, as the slow melting-down of the blister copper assists greatly in oxidizing the last traces of impurities without much slagging of copper), but the blister copper would lose its valuable quality of excessive "red-shortness" by which we are enabled to easily break the pigs apart, and other practical disadvantages would occur.

But if, in spite of these drawbacks, the oxidation of the copper is continued, the surface of the test becomes very convex, and finally it bursts, and, as the smelters say, "the worm crawls out," by which they mean that the test "spits," or has an outgrowth of molten copper on the surface, which looks like a worm, and is identical with the common "spitting" of a large silver button when cooled too quickly after cupellation.

Of course, as soon as the impurities in the blister copper are thoroughly oxidized, and indeed long before this point, the copper itself begins to oxidize rapidly, and to form a silicate of copper, taking up quartz from the hearth sides and fettling. I remember one instance in my early practice, when I had engaged a new refiner, and was called away just toward the close of the oxidizing stage, but anticipated no trouble, as the man brought good recommendations. When I returned in five or six hours I was informed that the copper was all sinking away and going to slag, as was indeed the case, the bath in the furnace having shrunk to infinitesimal dimensions, while the whole shed was filled up with blood-red slag. It seems that the new refiner had kept on oxidizing, expecting to see some peculiar appearance of the test which he was used to in refining the very impure copper that he had always worked on. Our copper being from a wet process, and of the highest grade of purity to begin with, failed



IMPROVED BRUECKNER'S CYLINDER, AS USED FOR ROASTING COPPER ORES.
Side Elevation.

Front Elevation.
(Fire-box removed.)

to show this peculiar phenomenon, which he termed "The bloomin' o' the Butt," if any one knows what that means, and he had patiently kept on oxidizing, regardless of the ominous lessening of the bath. The slag was all put back, with some charcoal and a little high-grade matte, which furnishes the best reducing agent that we have in the reverberatory furnace, and in the course of a few hours the copper was again obtained, and this time in a proper state of purity.

THE REFINING OF THE BLISTER COPPER IN GAS FURNACES AT ATVIDABERG, SWEDEN

The operation of refining at Atvidaberg is conducted in the same furnace that is used for the blister process.

I need hardly say that this is neither scientific nor economical, and is simply done on account of the small production and the fact that where labor and fuel are so cheap, and the capital of the owners is limited, it is considered better to delay the refining operation slightly than to go to the heavy expense of a separate refining furnace.

Of course, if there were much arsenic or other seriously injurious foreign substances in the blister copper, it would be practically impossible to produce a high grade of copper in this manner. But as the metal is remarkably free from the more dangerous impurities, this practice simply delays the refining process a few hours, as every atom of sulphur, iron, or other impurity resulting from the blister process, and which is certain to cling to the hearth and sides of the furnace in perceptible quantities, has to be oxidized and eliminated, either up the stack or in the slag, before refined copper of the proper quality can be produced. If it were practicable, it would be very much better to accumulate a large stock of blister copper before refining at all. Then to run the furnace solely on refined copper until the supply of blister is exhausted, by which time sufficient cement-copper and matte would have accumulated in stock for another blister campaign. The sole objection to this plan is the amount of capital it would take to pay the running expenses of the works and mines while the blister copper for a campaign was accumulating.

The charge of blister copper is very small, only about 5,000 pounds, and in a furnace in normal condition this is fused down in about four hours. The bath is then skimmed clean, the amount of slag being very small, owing to the unusual purity of the blister copper.

A light blast is turned upon the surface of the molten metal by means of a small tuyere introduced through a door opposite the working-door, and the copper is at the same time "flapped" constantly with a small rabble in the usual way. Of course the object of this stage is to expose the molten copper to the action of the air as rapidly as possible, and thus promote quick oxidation of the impurities that it contains, for the whole process of copper refining is based upon the fact that the impurities that are commonly contained in copper oxidize more easily and quickly than the copper itself. If they happen to possess a lesser chemical affinity for oxygen than the copper itself, they will, of course, remain unoxidized. A familiar example of such unoxidizable substances are silver and gold, which remain alloyed with the copper throughout the refining process, and indeed could, theoretically, be separated from the copper by oxidizing and slagging it off, leaving the precious metals behind, as in the cupellation of lead. In practice, this method has not been found economical, except under certain rare conditions, where it plays an important part in the separation of these metals.

I doubt very much if the air-blast from the tuyere striking the surface of the copper produces any very marked acceleration of the oxidizing process. I have more than once tried it under similar conditions in the ordinary English method of refining as practiced in this country, but could not see that it made any perceptible difference, and am inclined to think that it is merely a relic of the old-fashioned German refining method, where the tuyere was necessary to produce an air-current on the surface of the molten bath, there being no arch over the German refining hearth, and consequently no strong, natural air-current over the surface of the metal as there is where a reverberatory furnace is used.

But the degree of energy and skill with which the "flapping" is executed has a very marked effect upon the time necessary for this stage of the operation. It is most difficult, violent, and exhausting labor, and requires much tact and experience to strike the bath with the proper force and just at the proper depth to break it at that point into a shower of drops, which fall over a large area, and at the same time send out a large number of concentric wavelets of copper, that present a greatly increased surface to the action of the air-current that is hurrying over the surface of the bath to gain the chimney.

At Atvidaberg, this stage of the process usually occupies $1\frac{1}{4}$ hours, at the end of which period a test taken in the small ladle should show a very coarse, crystalline, light-red fracture, the small bubble in the center being a bluish-black instead of the reddish-brown hue that it showed during the early part of the oxidizing period.

The impurities are now nearly all removed, or else exist in the copper in the shape of oxide, while the metallic copper probably contains a considerable amount of suboxide of copper in solution.

The bath is skimmed clean, and the "poling" now begins, in order to lessen the amount of suboxide of copper (with which it was necessary to saturate the metal in order to thoroughly oxidize all foreign elements), as well as to reduce the foreign oxides dissolved in the copper. As these are reduced, they act in three different ways. Such as are at all volatile, pass off in gas. A considerable portion of the fixed oxides is taken up by the slag formed during the operation. While a third and very small portion is reduced back to metal and remain in the copper as impurities, iron being the principal of these.

The poling operation is divided here, as elsewhere, into two stages:

Dense poling and

Tough poling.

A large birch pole is thrust into the working-door, its butt being forced down into the molten bath by raising forcibly its thinner end which projects into the air, using the upper edge of the furnace door as the fulcrum, while the pole is retained in position by a piece of notched plank, set under its projecting end. The ebullition of gases is very great, and a rapid reduction takes place in the copper, while currents are established in every direction, so that all the metal in the furnace is rapidly brought under the influence of this powerful reaction.

The crystalline texture of the fractured test becomes of a much finer pattern, and the color much lighter and more pinkish, while the bubble in the center becomes rapidly smaller and more yellow, and soon disappears. When these reactions are fairly established, and the test is seen to take on a dense, fibrous texture from below upward, the stage of "dense poling" is considered at an end.

It usually takes about forty-five minutes, during which time the slag is skimmed several times, and as it now largely consists of a subsilicate of the oxide of copper which is too thin to skim off, sand is thrown on the surface of the bath to thicken the slag.

At the beginning of the "tough poling," a barrel of clean, selected charcoal is thrown upon the clean surface of the bath, and a fresh pole introduced. Tests are now constantly taken, the changes in the quality of the copper, though chemically very slight, being now both physically and commercially very important, and succeeding each other with extraordinary rapidity.

The fracture of the test is now becoming pinkish and fibrous, and minute pearly globules begin to appear under the magnifier, while the toughness of the metal increases in a remarkable manner.

As the process progresses, the even rows of fibers begin to disappear, and an irregular, very finely grained fiber replaces them, while the fracture looks more silky and the toughness increases.

In two or three minutes from this time, the pearls suddenly become distinct, the beautiful rosy-pink of refined copper appears, and the test becomes so tough that, after nicking it with a chisel, it can be bent back and forth many times in a vise without breaking.

The pole is now withdrawn, fresh charcoal is spread over the bath to entirely exclude the air, and the heated and clay-washed ladles are brought to the door.

The ladling of this small charge of 5,000 pounds or less took over an hour, having to be interrupted several times to pole the copper, and well exemplifying the claim I have so often made for large furnaces and large charges, that it is infinitely more difficult to handle small quantities of metal, as the slightest mischance, or undue splashing about of the copper by the ladles, causes oxidation, and gets it "out of set."

When we are ladling a proper amount, say 25,000 to 35,000 pounds of copper in one operation, we seldom have to put in a pole more than once or twice until we come nearly to the end of the ladling, and are dealing with something like the quantity that they begin with in Sweden.

And in order to be sure that there shall be no mistake in the quality of the copper, I have usually made a practice of closing the operation as soon as it becomes difficult to keep the metal "in set," and ladling out the little copper that remains in the furnace into sand beds, to be thrown back into the furnace at the next refining. There is no waste in this, and practically no expense, the sole drawback being the couple of hundred dollars invested in the extra amount of copper that is thus kept circulating in the process, and gets into market twenty-four hours later. With interest at 10 per cent. per annum, the cost of this practice would be about one-half cent per day, and allowing even 10 cents for the amount of fuel necessary to resmelt it (for the labor costs nothing, it being all done under contract, and the extra labor of throwing a few ingots back into the furnace not being heeded by the men), it will be seen that the cost is not to be considered in comparison with the very serious trouble and expense arising from sending out copper of unequal quality.

I mention these details, as I have heard this practice denounced as wasteful and extravagant, while I consider it a decided improvement, if a small one.

The refined copper produced at Atvidaberg amounts to about 90 per cent. of the blister copper charged into the furnace. Considering the unusual purity of this blister, the amount of slag produced is excessive, which arises from two causes already mentioned; one being the very small charge used, a large charge giving a much smaller per cent. of slag to be resmelted, while the other cause is the practice of using the same furnace for both the blister and refining processes, by which the latter operation is prolonged and more slag produced.

THE USE OF BASIC-LINED FURNACES FOR REFINING COPPER.

The remarkable results obtained by Mr. Percy C. Gilchrist in England from the use of basic material for hearths and linings of blister furnaces have been fully stated in the chapter on the blister process.

Encouraged by the unusual success that crowned his efforts, Mr. Gilchrist very naturally desired to apply his new ideas to the operation of refining copper.

As he states in his own paper on the subject, there is very much less chance for improvement in the refining process than in the blister process. For in the former, the impurities have already been mostly removed, and the amount of slag produced, even in ordinary furnaces, is very small.

Still he finds a most decided diminution both in the quantity and the copper percentage of the slag produced in basic furnaces during the operation of refining. And, as is quite as one would expect, he finds that the more impure the copper charged is, and consequently the longer the period of oxidation has to last in the refining furnace, the more useful does his basic lining prove to be. For on an acid hearth, the oxide of copper formed during the earlier stages of refining is constantly dissolving silica out of the hearth and lining, and augmenting the quantity of the slag.

He does not find, however, that the arsenic or antimony is removed any more quickly by the basic lining; rather the contrary.

The practical question immediately arises, whether it will pay to apply this method to our refining furnaces here in the United States.

My own opinion is, that under our present conditions the advantage gained would be so slight as to hardly pay for the change.

This is because most of the copper that with us undergoes a refining operation is already purer than average English copper is after the end of the refining process. The reason of this condition of things is partly, that nearly all matte produced in the United States that is at all arsenical or antimonial is also sufficiently rich in silver or gold to warrant the separation of these metals, which is, of course, accompanied by the elimination of the arsenic and the production of a copper of the very highest quality.

Still another reason exists, which I hesitated to mention in the first edition of this book. But feeling myself fortified with a few more years of experience and observation, which only serve to make me more certain of what I knew before, I no longer hesitate to say that the reason why our American copper market is in such a crude condition compared with that of England, the reason why our refiners hesitate to deal in the dry way with arsenical material, and above all, the reason why our grades of copper are divided with such charming simplicity into merely, "Lake copper," "Arizona copper," and "Montana copper," is that our consumers of copper are so ignorant of the peculiarities of different coppers, as well as of just what they need in the copper that they purchase for their own use, that many of them cannot form an independent judgment of the quality of a given brand of copper by physical and chemical tests, and therefore are obliged to frequently buy a better copper than they need to make sure of getting one that is good enough.

It is startling to English refiners to see our superb Lake copper used wholesale for making certain castings that would be just as good if an inferior metal were used, costing one to one and a half cents less per pound.

Having refined copper for the open market myself for several years, I can speak intelligently on this point, and in more than one instance have been unable to introduce a good, fair casting brand of copper, as the manufacturers who tried it assured me that their foreman was unable to obtain sound castings from it. The extreme sensitiveness of copper to external influences was well exemplified in these cases, for when, by permission of the owners, our own brand of copper was shipped to them in some old "Lake" barrels, its quality was found to be so improved by the packing that the foreman had no fault to find with it.

Until accurate physical and chemical tests are substituted for a mere glance at the color of the ingot or cake, and the manufacturers have learned to buy copper just good enough for their work and no better than they need, there is little encouragement for refiners to trouble themselves to improve the quality of their product.

So long as "Arizona" is bound to sell for a cent or more less than "Lake," regardless of how suitable it may be for the intended purpose, just so long will it be idle for the refiner to expend time and money in making a better quality of "Arizona."

But this crude state of affairs cannot last long, for Americans are not likely to remain indifferent to a matter that affects their profits so largely.

It is for the refiners to lead the way, and to urge upon the manufacturers the importance of adapting the quality of the copper that they buy to the purpose for which it is intended, and not use Lake copper with its superb malleability and high conductivity, for making castings.

When this matter is better understood, the inferior coppers will no doubt be much more generally used, and the present wide margin in price between them and Lake copper will be reduced to much narrower limits.

CHAPTER XVI.

TREATMENT OF GOLD- AND SILVER-BEARING COPPER ORES.

A VERY few words may serve to indicate the present practice in the separation of the precious metals from copper. The older processes employed for this purpose were by far the most complicated and wasteful operations known to metallurgy, and it is only since the discovery and introduction of the various "wet" processes that any but the richest coppers could be advantageously treated for the precious metals.

The Ziervogel process has only been successful in a few isolated cases, and demands such pure material and such skill in manipulation as to debar its use in ordinary instances, nor does it provide for the extraction of gold.

It is indisputable that the electrolytic methods are rapidly advancing to the front in the treatment of gold- and silver-bearing metallic copper, and have the great advantages of producing a copper of the best quality, but can only be profitably employed on matte carrying at least $30 per ton in precious metals and require a bulky and extensive plant.

The new Hunt & Douglas method, as applied to copper ores or mattes, seems to fill the gap more completely than any previous invention. By this method, the copper is extracted from the ore or matte after a very imperfect roasting, and being precipitated as a dioxide by sulphurous acid generated from pyrites, it is decomposed by about one-half its weight of metallic iron, the resulting cement being fit for immediate refining. The copper is obtained in a state of absolute purity even in the presence of arsenic and antimony, while the residues, containing every trace of the gold, silver, and lead originally present, may be smelted with lead ores in a blast-furnace. The process has long passed the experimental stage, and offers advantages peculiar to itself and unshared by any other.

The ease with which the small amount of gold sometimes present in cupriferous pyrites may be won is not realized by all copper smelters, although the method is extensively practiced in this country, as well as at Swansea and in Chili.

Owing to its great affinity for metallic copper, the gold contained in white metal may be concentrated into a very small bulk of the former by exposing the pigs of matte to a slow, oxidizing fusion, exactly as in the process for making blister copper. The operation, however, is interrupted as soon as a certain quantity of metallic copper is formed, when the furnace is tapped, and the product—now advanced to *pimple metal*, or even *regule*, from 82 to 88 per cent.—being examined, *bottoms* of metallic copper will be found under the first few pigs. This is the method pursued in making *best selected* copper; for not only does the small quantity of metallic copper extract the gold, but also the greater part of other foreign and injurious substances—such as arsenic, antimony, tellurium, tin, etc. The proportion of *bottoms* formed must vary with the quantity of gold present; in some instances, even a repetition of the processes being required to fully extract the more valuable metal. Silver is but slightly concentrated by this operation, as will be observed from the following assays made under the author's direction, want of space forbidding fuller details of this important process:

Assay of original white metal.		Proportion of bottoms formed.	Assay of bottoms.		Proportion thus extracted.		Assay of residual pimple metal.	
Gold.	Silver.		Gold.	Silver.	Gold.	Silver.	Gold.	Silver.
Ounces.	Ounces.	Percent.	Ounces.	Ounces.	Per cent.	Per cent.	Ounces.	Ounces.
0·64	93·3	6·4	9·60	213·4.	93·7	14·8	0·030	78·7
2·37	16·6	9·0	19·10	36·2	90·2	18·5	0·110	14·2
0·11	....	5·4	1·73	....	88·4	....	0·012	....

In examining this table, it must be remembered that a considerable concentration has taken place in the matte itself as well as in the copper bottoms, so that the results do not seem to agree; but the figures given are sufficient to indicate the general results of the process. Unless the furnace bottom is already well saturated with auriferous metal, a heavy loss in gold must be expected.

CHAPTER XVII.

THE BESSERMERIZING OF COPPER MATTES.*

While the reader must be referred to Professor Egleston's pamphlet, as well as to the literature of the future, for all details pertaining to this new and interesting innovation, the present treatise would be incomplete without a few remarks upon a process which promises to become of great importance when introduced where the conditions are suitable for its application. So many new and valuable improvements have been greatly injured and retarded by their miscellaneous and improper application that a few words of advice from one who witnessed the construction and starting of the first successful American copper-Bessemerizing plant may be of value. While there is no doubt of the technical success of the process as perfected by M. Manhès, and of the plant as constructed in this country under the direction of his pupils, it is only under certain conditions that its real usefulness can assert itself, and any attempt to apply it to all and every variety of circumstances would certainly result disastrously.

The Bessemerizing plant, as constructed at the works of the Parrot Silver and Copper Company at Butte City, and observed by the writer at its commencement, was adapted to two different duties: 1st. To receiving copper matte of a low grade—from 15 to 40 per cent.—and bringing it up to white metal—75 per cent.; or, 2d. To bringing white metal up to a very pure blister copper.

The impossibility of producing metallic copper from poor matte at one continuous operation is quite evident, as, aside from the difficulty of dealing with the great quantity of slag formed from the iron and other foreign bases contained in the low-grade matte, the amount of metallic copper produced therefrom would be too small for manipulation. For instance, the usual charge —2,000 pounds of a 20 per cent. matte—would yield less than

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* See *School of Mines Quarterly* for May, 1885, for paper on this subject by T. Egleston.

400 pounds of blister copper, a quantity far too small to submit to any blowing operation in a converter.

This is especially the case in M. Manhès's converter, where the tuyere orifices are situated at some distance above the bottom of the vessel. This latter peculiarity has been found an essential element of success; for, whereas in iron, the whole contents of the converter are homogeneous, the blast traversing the entire mass of metal, and oxidizing the impurities, which may be regarded as distributed equally throughout the molten iron, so that the whole product gradually becomes pure without any division into a finished and a still uncompleted portion—in treating copper matte above 75 or 80 per cent., the liquid metal that until that point has been homogeneous throughout, must then begin to separate into two portions—namely, sulphide of copper, and metallic copper that has been deprived of its sulphur by oxidation. As the process continues, the latter product augments in quantity, while the former decreases, until the last atom of sulphur is removed. Were the tuyeres at the bottom of the converter, the metallic copper would soon chill and obstruct them, and it was not until M. Manhès raised them to such a height as to allow the quiet subsidence of the metallic product below their inlets, that he attained complete success. It is necessary, however, that a sufficient amount of copper be present to support the superincumbent layer of liquid matte above the tuyere openings, so that the blast may traverse a molten and oxidizable product to the last, and thus generate sufficient heat to maintain the entire mass in a liquid condition.

Experience has shown that upon introducing a matte containing from 72 to 75 per cent. of copper into the converter, the process advances with great rapidity and completeness, while a matte of 60 or 65 per cent. requires several times as long for its oxidation. On the other hand, a low-grade matte of even 15 or 20 per cent. advances with satisfactory speed to the condition of white metal—from 70 to 75 per cent.—and there stops or continues very slowly. The practice has, therefore, been adopted of interrupting the oxidation of the low-grade matte at the point indicated, pouring it out of the converter, that it may separate from the slag, and subsequently completing the process in a second vessel, the products of two or more "blowings" of poor matte being united to form a single charge for blister copper.

It is therefore necessary for economy to have two sets of converters, and while three converters are required for a single operation, five, or possibly four, converters are sufficient for the complete process. A converter will usually stand from eighteen to twenty-four blows (twenty-four hours) without repairs, so that for the single operation, one converter is undergoing repairs, the second is drying, while the third is in use.

In France, a separate cupola is used for melting the matte for the converter; but the Parrot Company has found it feasible to run the matte directly from the ore blast-furnace to the converter.

No fuel is required to keep up the temperature in the converter while working on low-grade matte; but the operation for blister copper requires the occasional use of a few pounds of coke to keep up the necessary high temperature.

While the construction of the converter plant is simple, the management of the same requires much care and experience.*

The appearance of the flame issuing from the mouth of the vessel is of little value as a guide, owing to its changeable color from the various foreign constituents of the matte. The tuyeres require constant opening with an iron rod, taking one man's whole time.

The lining of the boiler-iron converter is of crushed quartz (or pure siliceous sand), mixed with enough plastic fire-clay to hold it together. It is rammed in large balls, the original shape and size of the interior vessel being obtained from an oil barrel, used as a core, about which the lining is rammed. The same material is used for repairs. A cylinder blowing-engine supplies the blast, which is much less powerful than in ordinary Bessemer work, the height of the liquid column of metal being only a few inches, and the entire charge not exceeding 2,200 pounds.

Any attempt at estimating the saving effected by this operation under any given circumstances would be futile, as the process, although satisfactory to its owners, and thoroughly successful in the opinion of the author—speaking as a spectator—is still under constant improvement, and when stripped of its crudities and adapted to American conditions will give very different results from those obtained at its first introduction.

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* The Parrot Company's plant was erected by pupils of M. Manhès.

There is every reason to believe that its capacity will be greatly increased, and, as even in its present state it can show a great saving in fuel and labor above any of the older methods, there is little doubt that it will, ere long, be recognized as an essential feature of every large copper plant, except where very cheap fuel or other peculiar conditions neutralize its advantages. The elimination of arsenic and antimony by this operation was highly satisfactory, as far as the author's observations extended. Whether an undue loss of silver by volatilization may also occur in argentiferous mattes, yet remains to be decided. The slags from the Bessemerizing of low-grade mattes form a welcome basic flux in the ore-furnace, while the lining of the converter is partially protected from their corrosive influence by the feeding of pulverized siliceous ores through the tuyere-holes with the blast.

Latest advices from the President of the Parrot Company report great improvements in the capacity and economy of the process.

A simple set of converters now produces 50,000 pounds of 99 per cent. copper, daily, from 60 per cent. matte, and casts of 2,400 pounds of pig-copper are now made from a single charge of matte of this grade.

Since the publication of the above the Parrot works have enlarged and improved their Bessemer plant, but no statements of cost and results can be obtained from them, and the method has not gained ground in this country.

THE END.

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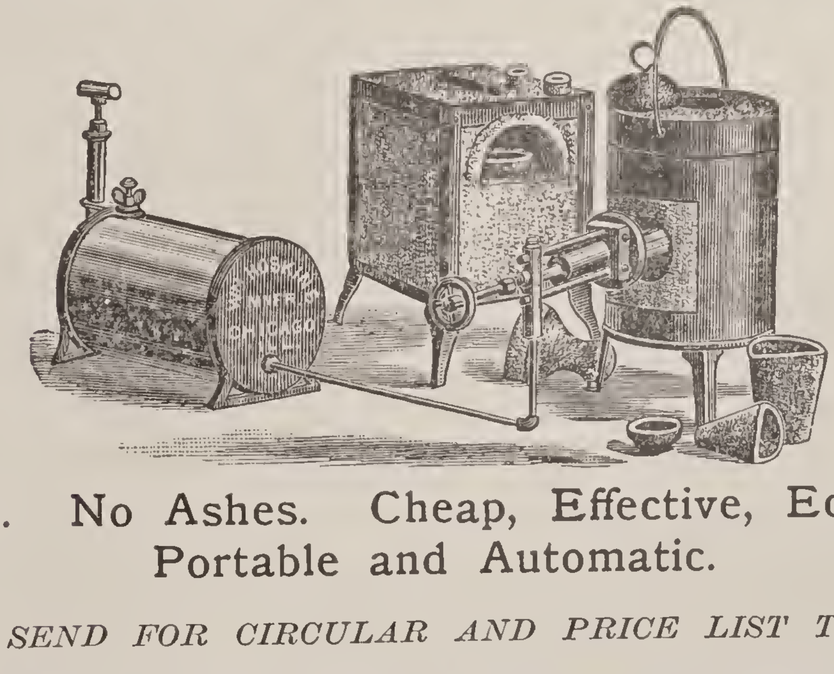

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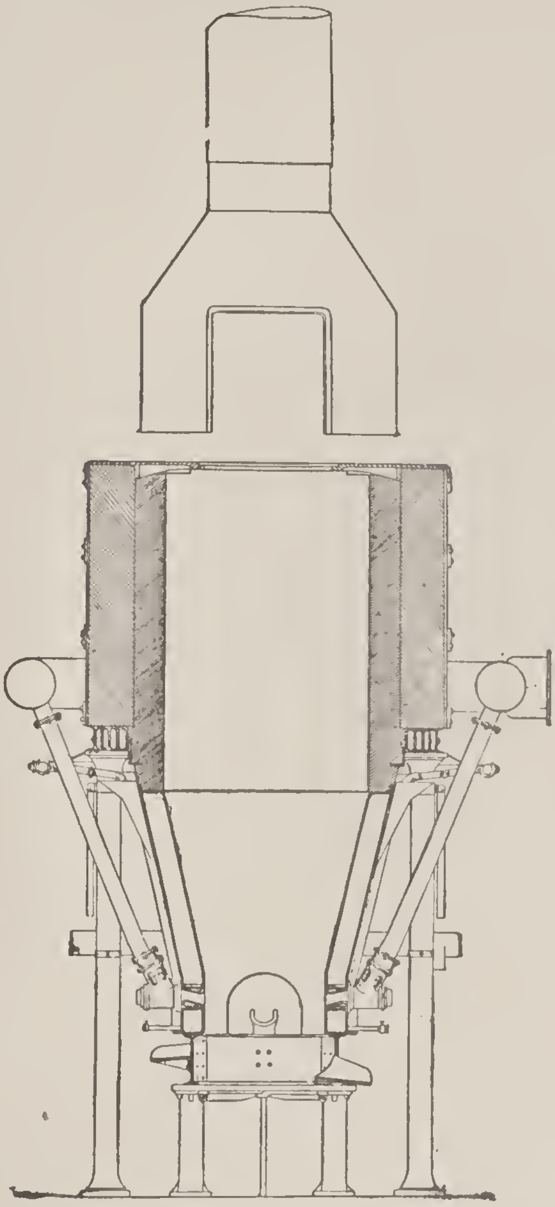

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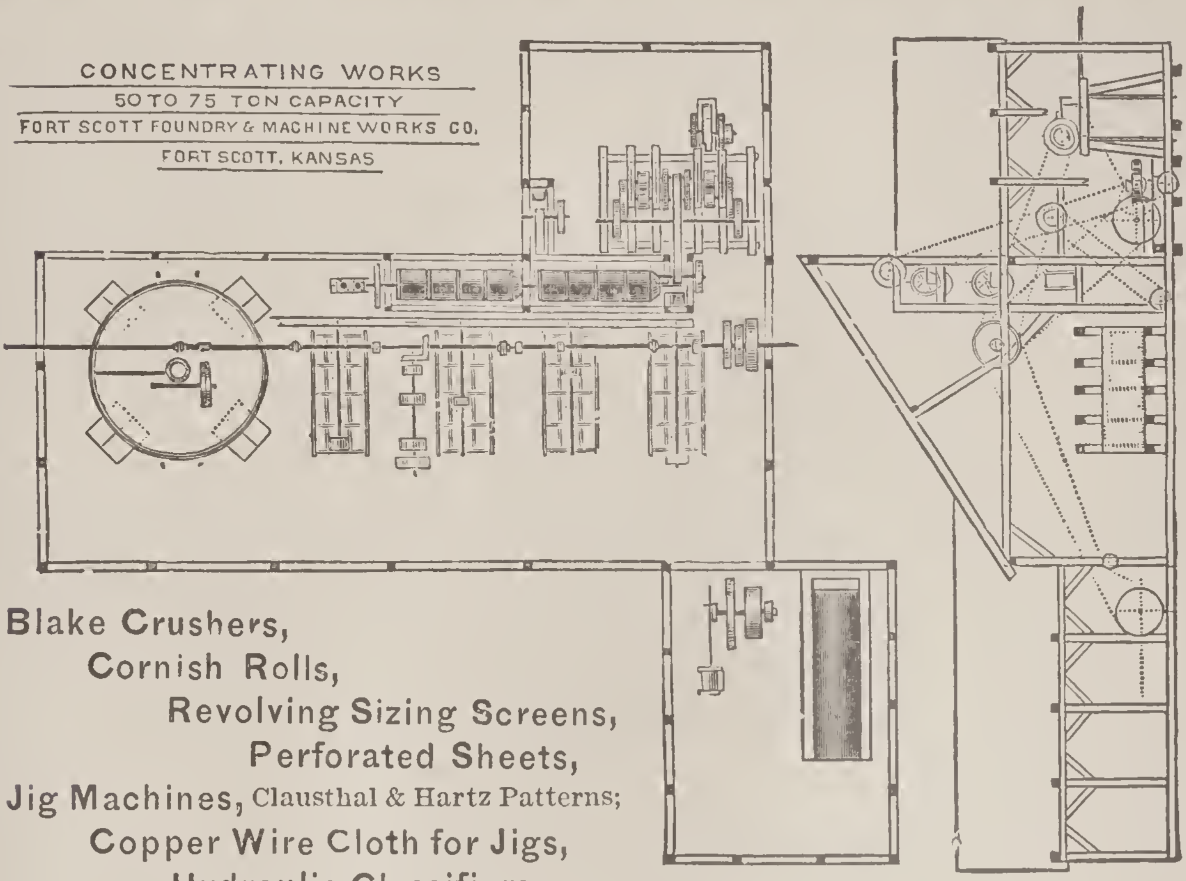

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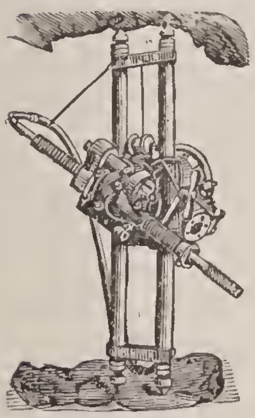

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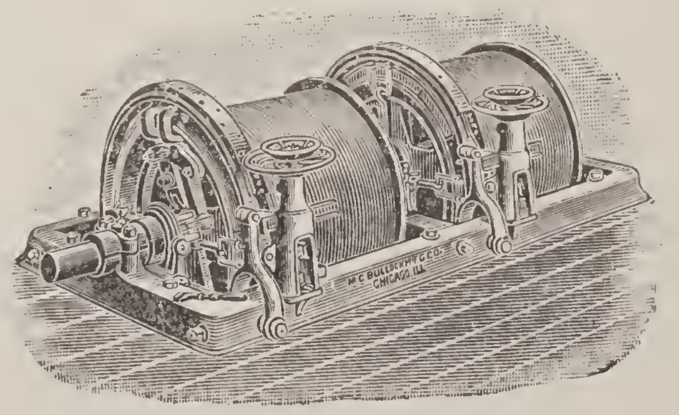
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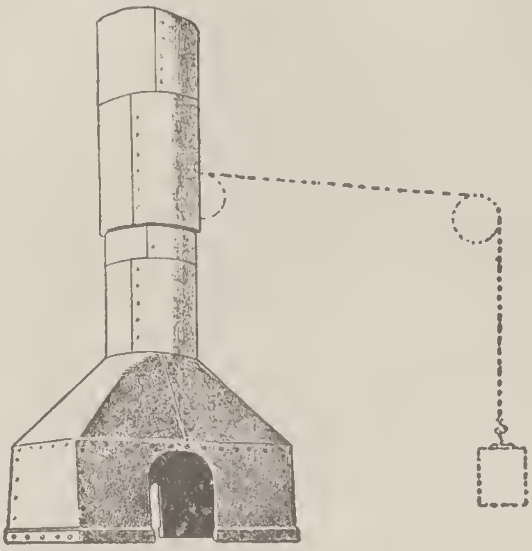

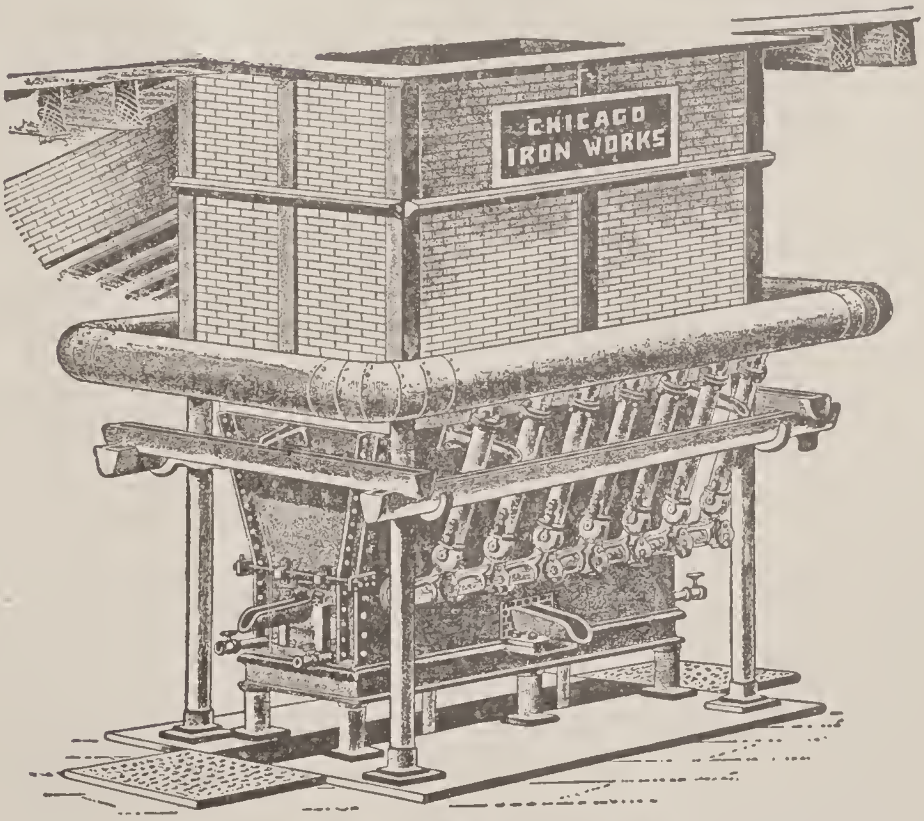

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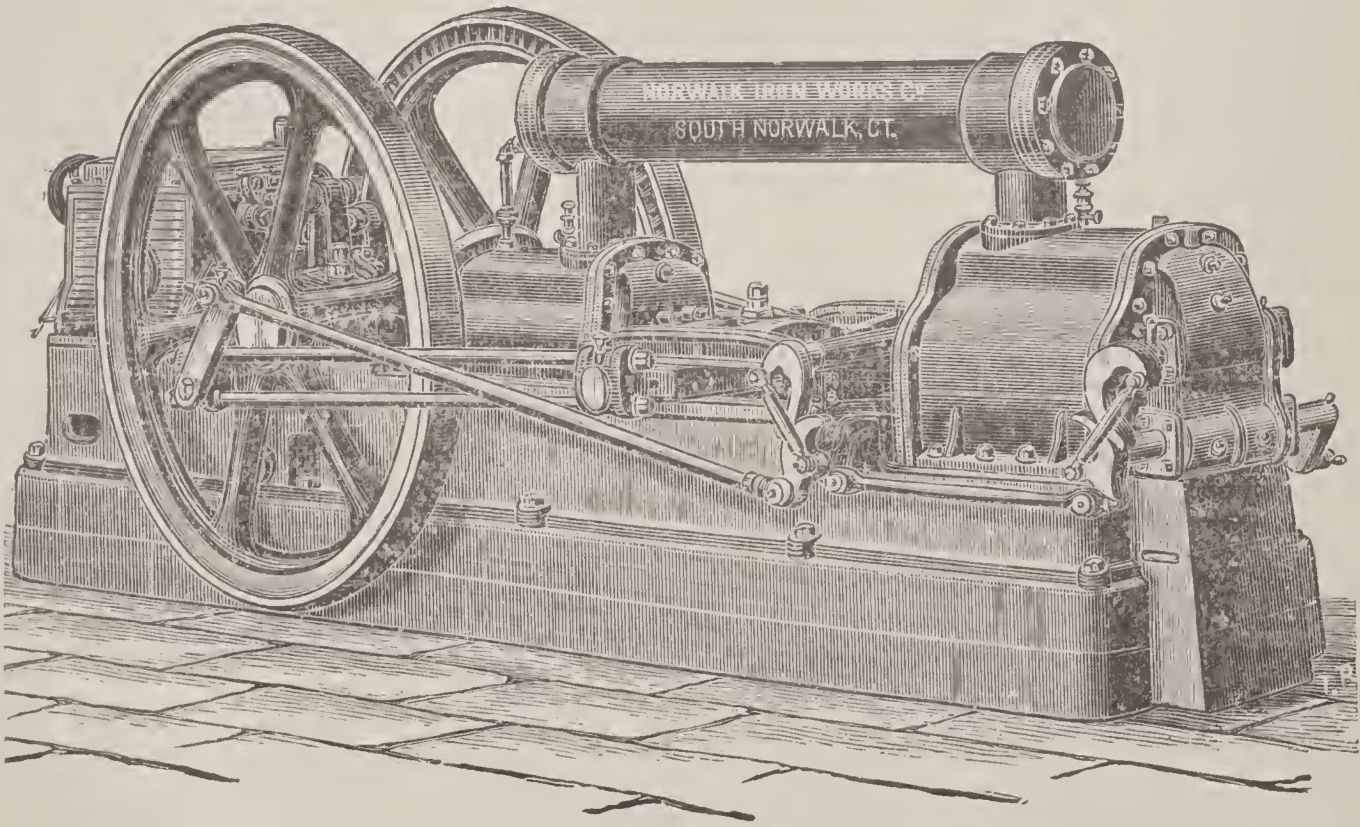

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